PRACTICAL COAL MINING

PRACTICAL COAL MINING:

A MANUAL FOR MANAGERS, UNDER-MANAGERS, COLLIERY ENGINEERS, AND OTHERS.

BY

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WITH NUMEROUS PROBLEMS ARISING FROM COLLIERY WORK

AND

523 Figures and Diagrams.

FOURTH EDITION, ENTIRELY REVISED, RESET THROUGHOUT, AND ENLARGED.

LONDON:
CHARLES GRIFFIN & COMPANY, LIMITED,
EXETER STREET, STRAND.
1905.

PREFACE TO THE FIRST EDITION.

NOT many years ago the works on Coal-Mining were few, and in most instances so expensive as to be beyond the reach of the ordinary student or practical miner. This state of affairs has been to a large extent remedied of recent years by the issue of several works of more moderate dimensions and price. Between the small elementary text-book and the still large and comparatively costly work of reference, however, there yet remains a considerable gap, which it has been the Author's endeavour in this volume to fill. The best authorities have been freely consulted and, with due acknowledgment, laid under contribution; while the latest methods of working and the most modern machinery have been described, with the object of presenting an up-to-date account of the important industry under consideration.

Since the publication of the excellent treatise by Jonathan Hyslop, thirty years ago, no text-book dealing to any extent with Scottish practice has, so far as the Author is aware, been published. As his experience has been gained largely in Scotland, he has attempted to remedy this omission, and it is to this that the occasional occurrence of a few Scotch words or phrases must be attributed, although these have been avoided, as far as possible, when reference is made to methods prevailing elsewhere. Less attention has been paid to literary style and elegance than to the production of a thoroughly practical and plainly worded text-book, designed not only to aid those endeavouring to qualify themselves for positions as colliery managers and other responsible officials, but also as a daily guide and reference-book for all engaged in and about the Colliery.

The Author gratefully expresses his indebtedness to other sources, which, except in cases where a difficulty in tracing the authorship

was experienced, he has acknowledged in the text. The works of Hughes on Coal Mining, and of Foster on Ore and Stone Mining, have been quoted, and a few of the illustrations have been borrowed from those works.

The Author has further to acknowledge the kind assistance of Mr John Dodds, who helped to prepare many of the drawings for the work. He also desires to express his thanks to the publishers for the pains they have taken both as regards the text and the illustrations.

G. L. KERR.

BO'NESS, N.B., *September*, 1900.

NOTE TO FOURTH EDITION.

IN issuing this new edition advantage has been taken to thoroughly revise and extend the work. Over forty pages of new matter have been introduced, and the whole rearranged. A considerable number of new illustrations have been added, while at the same time a number included in former editions have been deleted owing to their having become obsolete in modern mining practice. With the additions and improvements that have been made in the present edition, the author hopes that its usefulness will be correspondingly increased.

G. L. K.

GLASGOW, *October* 1905.

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PRACTICAL COAL-MINING.

CHAPTER I.

THE SOURCES AND NATURE OF COAL.

Introductory.—The art of mining and science of geology are so closely related that it has become almost impossible to write a treatise on the former without referring to the latter. Text-books on coal-mining are therefore usually prefaced by a short introductory chapter on geology, and the present volume will, in this respect, conform to established custom.

The earth is composed of mineral matter in various combinations which are included under the general term of *rock*. It is an oblate spheroid in shape—that is, a sphere which has been flattened at the poles. It was at one time supposed to consist of a hard, solid, outer crust 10 or 12 miles thick, and an interior of molten material at a very high temperature. This theory was deduced from the increase in temperature observed in subterranean workings, and from the fact that molten lava is thrown out by volcanoes during eruption; but, according to Lord Kelvin, it is much more probable that the earth is a rigid mass from surface to centre with the properties of a solid.

Definition of the term Rock.*—"A rock may be defined as a mass of matter composed of one or more simple minerals, having usually a variable chemical composition with no necessarily symmetrical external form, and ranging in cohesion from mere loose débris to the most compact stone. Granite, sandstone, mud, peat, etc., are all recognised as rocks."

Division of Rocks.—Rocks are divided into three classes, viz., aqueous, igneous, and metamorphic; or into two, stratified and unstratified.

Aqueous rocks are those which have been deposited where we now find them, by the agency of water. They are generally in layers or beds lying parallel to each other, and are often termed sedimentary

* *Text-book of Geology*, p. 57, by Sir Archibald Geikie.

rocks or deposits. *Igneous rocks* are those which have been subjected to the action of heat and retain no traces of stratification or bedding. *Metamorphic rocks* are rocks in which a crystalline rearrangement of the materials has taken place. They are sometimes called *altered rocks*. Marble is one of the best and most representative specimens of a rock of this class.

Aqueous or *sedimentary* rocks are deposited in definite layers or beds, this arrangement being termed *stratification*. When the deposits form very thin layers, such as occur in shale, they are said to be *laminated*.

Cleavage.—Cleavage is the term applied to the tendency of rocks and minerals to split along certain planes other than those of stratification, which occurs in stratified rocks and which tends to break the rock up into more or less cubical blocks. Generally when a rock is much intersected by cleavage planes, it loses its property of splitting along the bedding planes. Cleavage planes are said to be due to great pressure.

Inclination of Strata.—The strata which compose the crust of the earth were no doubt deposited in horizontal layers; but only limited areas are now found in that position. In all parts of the world beds of rock are usually inclined at a greater or less 'angle of dip' to the horizon, hence they usually come to the surface at some point, and when this happens it is termed the *outcrop* of the bed. In flat, low-lying stretches of country few outcrops may be seen, while in hilly country, especially where the district is intersected by ravines and river-courses, the strata may be seen to outcrop frequently. It is in such positions that rocks can be most easily and advantageously studied.

Dip.—If a bed of rock be inclined to the horizon it is said to *dip*; the point of the compass to which it inclines is called the *direction of dip*, and the angle, or degree of deviation, which the strata make with the horizon is termed the amount, or *angle of dip*. This angle is expressed in degrees, and is usually measured by an instrument called the clinometer.

Strike.—The prolongation of the strata in a line of bearing at a right angle to the dip is called the strike. Thus, if the dip be due north and south, the strike will be due east and west. Sometimes the strike and outcrop coincide, as in the case of vertical beds, but more usually it varies with the stratigraphical contour of the beds.

Anticlinal and Synclinal.—In many parts of the world the strata are contorted and bent into folds. The French and Belgian coal-fields furnish examples of such distortion. Where the strata dip away from an axis so as to form an arch or saddle, they are termed '*anticlinal*.' Where they dip towards an axis, forming a trough or basin, they are called '*synclinal*' (see fig. 1).

Dislocations, Faults, and Dykes.—Rocks are liable to many disturbances and to fracture. Such fractures may be either simple

fissures or rents in the rocks without any displacement on either side (*joints*), or the strata may have a greater or less amount of displacement (*faults*). In the largest proportion of cases there are both fracture and displacement in the beds, the rents becoming both 'fissures' and 'faults' (see fig. 2). Faults may vary in width from

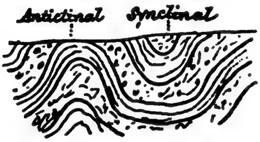


FIG. 1.

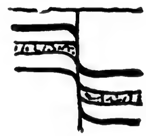


FIG. 2.—Fault.

mere sharply defined lines with very little displacement up to gaps of many yards in width and hundreds of yards displacement. All the British coal-fields are traversed by many of these faults, the main faults in nearly all cases running almost due east and west. Sometimes, however, faults branch off or run into one another. The inclination which a fault makes with the vertical is termed the *hade*, and the line of fracture the *vees* of a fault. The degree of vertical displacement is known as the *amount of throw*. Faults are sometimes vertical, but are generally inclined. The largest faults, *i.e.* those which have the greatest vertical displacement, commonly slope at high angles. Those of only a few feet displacement may be inclined at angles as small as 18° or 20° from the horizon, but this is exceptional.

A fault is termed a down-thrown fault if the observer is looking from the higher to the lower level of displacement; and an up-throw fault if in the opposite direction. Faults may be either *Normal*, *Reversed* (fig. 3), or *Overlap*, *Trough* (fig. 4), or *Step Faults*. When

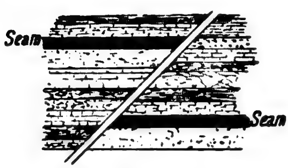


FIG. 3.—Reversed Fault.

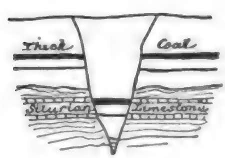


FIG. 4.—Trough Fault.

parallel faults run in the direction of the strike, and have their down-throw with the dip, this tends to prevent certain outcrops from appearing. On the other hand, a succession of step faults, with down-throws against the dip, may cause the same beds to crop out again and again, and hence be mistaken for a number of different seams. When a bed has, from some cause, had a portion denuded or worn away, it is known as a *dumb fault*, or *wash-out*. The actien of water is the most frequent cause of a wash-out.

The term *dyke* is often confused with the term *fault* by miners, and taken to mean the same thing.

When a fault occurs there is displacement of the strata; but a dyke is usually unaccompanied by any displacement. Dykes are wall-like masses of rock which traverse strata in succession from unknown depths and appear in many instances at the surface. They usually consist of basalt or allied rock, and are of volcanic origin, having apparently been 'intruded' while in a liquid state into fractures. Dykes affect very materially the quality of the coal intersected by them, which is '*burnt*' into a soft, cindery, and sooty state, or altered into a hard and incombustible substance. The distance that the coal-seam is affected on either side of a dyke is usually about two-thirds of its width. Dykes vary in width from less than a foot to 70 ft. and upwards. It is not unusual for them to run in nearly straight courses for many miles. Sometimes they occur along the lines of a fault, but very often they are unconnected with faults in any way. Frequently they are found to cross faults without being in the least deflected out of their course thereby.

Division of Rocks into Groups.—The rocks forming the crust of the earth in the British Isles have been divided into five main groups. They are disposed in the following order:

Quaternary or Post-Tertiary	Recent and Pre-historic. Pleistocene.
Tertiary or Cainozoic	Pliocene. Miocene. Oligocene. Eocene.
Secondary or Mesozoic	Cretaceous. Jurassic. Triassic.
Primary or Palæozoic	Permian. Carboniferous. Devonian and Old Red Sandstone. Silurian. Cambrian.
Archæan or Azoic	Primitive Schists. Gneiss and other Crystalline Rocks.

Coal-measures.—The formation which has the greatest interest for the coal-miner in this country is the Carboniferous, for it is in this formation that coal is found most abundantly. It consists of three divisions of strata, viz. :—Coal-measures, Millstone Grit, and Carboniferous Limestone.

The upper division or Coal-measures are the strata where coal-seams are most abundantly found. In the English coal-fields, very few workable seams are found below these measures. The millstone grit is usually composed of coarse yellow sandstones, flagstones, shales, and a few thin seams of coal. In the Scotch coal-fields valuable seams of coal and ironstone are found below the

millstone grit. In the Carboniferous formation of Scotland, the following four divisions are made :—

Coal-measures	Consisting of (*a*) an upper series of red and purple sandstones and shales enclosing thin coal-seams, and (in Ayrshire and Fifeshire) thin limestone bands ; (*b*) the productive coal-measures, consisting of white and grey sandstones, shales, coals, fireclays, and ironstones, but no limestone.
Millstone Grit or Moor Rock	Coarse thick sandstones with shales, fireclays, thin seams of coal, clayband ironstone, and, occasionally, beds of limestone.
Carboniferous Limestone	This series is often sub-divided into (*a*) thick sandstones and beds of shale with three varieties of limestones and some coals ; (*b*) a group of ordinary coal-measures very similar to the upper coal-measures, and containing valuable seams of coal and ironstone, but no limestone ; (*c*) sandstones, limestones, shales, with some coals and ironstones.
Calciferous Sandstone	This series may be sub-divided into (*a*) the upper group, consisting of sandstones, shales, oil shales, some very inferior coals, ironstones, and limestones ; (*b*) the beds lying below, extending to the base of the carboniferous rocks.

It will be seen from the above that the coal-measures in Scotland differ greatly from those of England, inasmuch as a large proportion of the coal in the Scotch coal-fields is found below the millstone grit, while in the English coal-fields very few valuable seams are met with below that formation.

Coal found in other Formations.—It has been shown that coal is most abundantly found in the Carboniferous strata, but it is not entirely confined to that formation, being often found in others—although such coal is seldom of much value compared with that of the coal-measures—both above and below the Carboniferous formation. In New South Wales coal is got from the Devonian series of rocks; at Bonn, in Germany, and at Bovey Tracey, in Devonshire, the coal-beds are, presumably, of Miocene age ; the brown coals of New Zealand and Australia are believed to be late Tertiary deposits, while at Brora, in the North of Scotland, coal is found in the Jurassic formation. Nearly all these are, however, lignites as distinguished from the true coals found in the Carboniferous formation.

Rocks and Minerals associated with Coal.—Coal when found is generally associated with sandstones, shale or blaes—known amongst miners as bind—limestone, fireclay, ganister, ironstones, iron pyrites, and in Scotland, oil-shales. Most coal seams rest on a bed of fireclay ; in some districts this under-bed takes the form of ganister.

Formation of Coal-fields.—It is a noticeable characteristic of coal-fields that they take the form of a basin, dipping from all sides towards a central axis. Hence we get the seams cropping out frequently at the surface, which allows of large areas being easily reached and worked. Were it not that coal-fields assume this shape a large part of our coal supply would inevitably be found at too great a depth to be workable.

Origin of Coal.—Regarding the origin of coal numerous theories are held, one being that it was formed in the position in which we now find it, and is the product of vast forests which grew, flourished, and decayed on the site of our present coal-fields. Another is that it was brought into its present position by what is termed the 'drift' process, the forests being supposed to have flourished and decayed in one part of the world, while the accumulated 'humus' was swept into its present position by the agency of water or ice. While the 'drift' theory may explain the origin of a few isolated deposits, there is little doubt that the first mentioned theory is correct.

Definition of Coal.—Coal is a substance which it is easier to recognise than to define. Nearly everybody is familiar with the appearance and uses of this common mineral, but its definition is attended with several difficulties.

Dr Percy defines it as: "A solid, stratified, mineral, combustible substance, varying from dark brown to black, opaque, except in extremely thin slices, brittle, not fusible without decomposition."

Sir Archibald Geikie defines coal as: "A compact, brittle, velvet-black to pitch-black, iron-black, or dull, sometimes brownish rock, with a greyish-black or brown streak, and in some varieties a distinctly cubical cleavage, in others a conchoidal fracture. It contains from 75 to 90 per cent. of carbon and a small percentage of sulphur generally combined with iron. It has a specific gravity of 1·2 to 1·35, and burns with comparative readiness, giving a clear flame and a strong aromatic or bituminous smell, some varieties fusing and caking into cinder, others burning away to a mere white or red ash." Or more shortly: "Coal is composed of compressed and mineralised vegetation."

Classification of Coal.—The varieties of coal may be classified as—

(1) Anthracite, or smokeless coal.
(2) Steam, free burning, or dry coal.
(3) Bituminous, or caking coal.
(4) Cannel, parrot, or gas coal, including the Boghead variety sometimes called Torbanite.
(5) Lignite, or brown coal.

All these varieties have had a common origin; they are all accumulations of ancient vegetation which has undergone chemical change under certain conditions. In the 'lignite' or 'brown' coal this change has been less complete than in the others.

Anthracite, sometimes also called 'blind-' and 'stone-' coal, has usually a brilliant black lustre, breaks with a conchoidal fracture, and does not soil the fingers when handled. It has been supposed that it has, at some period, undergone a sort of natural coking process, under the influence of subterranean heat, and that this has driven off a large proportion of the hydrogen, oxygen, and nitrogen it originally contained. Anthracite gives off little or no smoke and is difficult to ignite, but when burning gives out intense heat. It

consists almost entirely of carbon, the best qualities containing 90 to 95 per cent. with 5 to 10 per cent. of hydrogen, oxygen, and nitrogen.

Steam coal closely resembles bituminous coal, from which it differs only in being slightly harder, lighter, and more compact. It does not cake when heated, however, and it is practically smokeless. Its specific gravity varies from 1·27 to 1·30. On analysis it yields approximately 89 per cent. of carbon, 4·5 per cent. of hydrogen, 3 per cent. of oxygen and nitrogen, and 3·5 per cent. of ash.

Bituminous coal, also known as '*free burning*,' '*smoking*,' or '*flaming coal*,' when ignited, burns readily with a yellow flame, giving off smoke freely. On heating it swells into a pasty, bitumen-like mass which ultimately becomes solid. Bituminous coals are misnamed, as they contain no true bitumen. There are several varieties of bituminous coal, which are distinguished according to their mode of burning, which depends chiefly on the relative proportions of carbon, oxygen, and hydrogen they contain. Steam coal approaches anthracite in its properties. Dry or non-caking coal is another variety; it does not possess the property of caking which makes coal so valuable for household purposes. Non-caking coals are generally hard and compact, and when in a fine powdery state do not cohere when heated.

Cannel is generally classed as a variety of bituminous coal, although it is not a true coal, as it contains a certain amount of argillaceous matter, and sometimes even passes into shale or ironstone. Cannel or *gas coal* differs a good deal in appearance from ordinary bituminous coal, being of a dull, lustreless, black colour, not splitting readily into thin layers, and generally devoid of vegetable structure under the microscope. The best qualities of cannel are of a tough nature and can be cut readily with a knife; ornaments are frequently made from cannel of this kind. An analysis showed—

Volatile matter (containing ·58 of Sulphur),			40·28 per cent.
Coke, consisting of	Carbon,	49·40	56·22 ,,
	Sulphur,	0·29	
	Ash,	6·53	
Water, expelled at 212° Fahr.,			3·50 ,,
			100·00

Cannel coal contains a comparatively large percentage of oxygen and hydrogen, and it is therefore valuable for the manufacture of coal gas or paraffin oil, and is only distinguished from the bituminous shales now so extensively used in the manufacture of paraffin by the much smaller proportion of ash which it contains. The yield of gas from cannel coal varies from 10,500 to 13,500 cubic feet per ton, and from bituminous coal, from 9000 to 10,000 cubic feet.

Boghead, or *torbanite*, is a mineral occurring at Boghead, near

Bathgate, in Scotland. It has long since been practically exhausted. The mineral was brownish-black, and had a specific gravity of about 1·15. It contained 63 per cent. of carbon, 9 per cent. of hydrogen, 20 per cent. of ash, and 8 per cent. of oxygen and hydrogen. It yielded 15,000 cubic feet of gas, and about 70 to 80 gallons of oil per ton.

Lignite or Brown Coal.—'Lignite' or 'brown' coal is the term usually applied to deposits of more recent origin than coals found in the carboniferous formation, to which formation true coal belongs. Lignites vary in colour from a light earthy brown to a deep lustrous black, undistinguishable from ordinary bituminous coal. They contain 50 to 70 per cent. of carbon.

In New Zealand the coal worked is of the lignite variety, and is not of a very high quality.

Brown coals proper usually contain a larger percentage of carbon and a smaller percentage of oxygen than the true lignites.

The general composition of lignites and brown coal may be seen from the following analyses :—

	Carbon.	Hydrogen.	Oxygen and Nitrogen.
Lignite from Bovey Tracey,	67·9	5·8	26·3
,, ,, Cologne, .	67·0	5·3	27·7
Brown coal from Hungary, .	72·5	5·4	22·1
,, ,, ,, Tasmania,	71·9	5·6	22·5
,, ,, ,, Auckland,	72·2	5·4	22·4

The following table shows the various changes through which coal passes during its transition from wood to anthracite :—

	Weight of 1 cubic foot in lbs.	Carbon per cent.	Hydrogen per cent.	Oxygen and Nitrogen per cent.
Wood, average, ...	30	50·29	6·09	43·62
Peat, ,, ...	50	60·83	5·89	33·28
Lignite, ,, ...	70	67·43	5·59	26·98
Brown coal, average, . . .	75	72·92	5·4	21·58
Bituminous coal, average, . .	80	83·48	5·34	11·18
Anthracite, average, . . .	90	95·35	2·47	2·18

Selection of Coal.—Dr Percy says that the only sure guide in the selection of coal for any purpose is to make a practical trial on a large scale. A good deal of information may, however, be obtained from reliable chemical analyses, but as a rule the thermal value of a fuel, as determined by a physical test, is never even approximately realised in practice. An anthracite coal with no gas or flame may be suitable for one purpose, while a bituminous coal full of rich smoky gas may be most economical under other conditions. In any fuel large quantities of ash are objectionable, as they reduce the quantity of available combustible material per ton of fuel, and increase labour in handling both fuel and ash, while the fires require more frequent cleaning,

which entails a reduction in the efficiency of the boiler by the chilling influence of the cold air admitted during the process.

A large percentage of moisture is also objectionable, as a portion of the calorific power of the coal is unproductively expended in evaporating the combined water. The following analyses show the composition of the two varieties of good burning coal :—

	Caking Coal.	Non-caking Coal.
Carbon, .	75 per cent.	76 per cent.
Hydrogen, .	4 ,,	4·3 ,,
Oxygen, .	16 ,,	16 ,,
Ash, . .	3 ,,	3·3 ,,
Water, .	6·3 ,,	5·4 ,,

Coals are also sometimes classed as High Quality and Low Quality coal, as shown by the following analyses * :—

	High Quality Coals.			Low Quality Coals.		
	Anthracite.	Semi-bituminous.	Bituminous.	Anthracite.	Semi-bituminous.	Bituminous.
Fixed carbon, .	88·5	75·5	53·5	75·0	67·0	46·5
Volatile matter, .	5·0	18·0	40·0	5·0	15·0	34·0
Sulphur, . .	0·5	0·5	0·5	2·0	3·0	3·5
Ash, . . .	5·0	5·0	5·0	16·0	12·0	12·0
Water, . .	1·0	1·0	1·0	2·0	3·0	4·0

Calorific Power of Coal.—Modern requirements now demand the most economic generation of heat for a given expenditure of fuel, no matter to what purpose the coal is put. As already stated, the thermal value of fuel is not easily ascertained with any high degree of accuracy by chemical or physical tests, and only approximate values can be looked for.

The simplest method of finding the calorific power is by using the instrument know as Thomson's calorimeter, which is largely adopted for this purpose. The principles upon which the test is based are :—(1) That the latent heat of steam is equal to 967° Fahr., and (2) that coal or other fuel burned in pure oxygen evolves the same amount of heat as when completely consumed in atmospheric air. The test is carried out as follows: A measured weight of fuel is dried, finely powdered, and intimately mixed with the necessary proportions of a mixture consisting of three parts of potassium chlorate and one part of potassium nitrate. This mixture, which will burn freely without a supply of air, is placed in a copper cylinder *b* (fig. 5), which is primed with a fuse. This cylinder is placed within the copper combustion vessel *c*, and is then immersed in a glass jar *a* containing a known weight of water. The fuse, in the small cylinder *b* containing the chlorate mixture, is lighted, and the appliance is plunged into the glass cylinder containing the water, and is covered by the

* *Practical Engineers' Pocket Book*, 1897, p. 324.

second copper cylinder c, the cock at d being shut. After a few seconds the fuse ignites the mixture of coal and potash, and the products of combustion, passing through the water in a finely divided state, communicate the whole of their heat to the water. The temperature of the latter is carefully noted at the commencement and end of the test, and it is only necessary to multiply the weight of water by the number of degrees of heat communicated to it to find the calorific value of the fuel. The amount of water capable of being converted into steam per pound of fuel burnt is directly as the elevation of the temperature; thus, if the thermometer showed a rise of 7·5°, then *one pound of fuel would evaporate* 7·5 *lbs. of water.* Tables to facilitate calculation are supplied with each instrument.

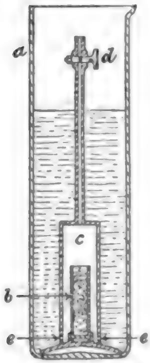


FIG. 5.—Thomson's Calorimeter.

The theoretical evaporative power of fuel may also be calculated from the ultimate analyses by the formula:—

$$P = 15\left\{c + 4{\cdot}28\left(H - \frac{O}{8}\right)\right\}$$

Where C = weight of carbon in 1 lb. of fuel.
H = ,, hydrogen ,,
O = ,, oxygen ,,
P = lbs. of water at 212° F., converted into steam at 212° F. per lb. of fuel.

As an example, in the analyses of the caking coal already given the carbon was 75 per cent., the hydrogen 4 per cent., and the oxygen 16 per cent. These would be in the ratio of ·75, ·04, and ·16; then by applying the above formula we have—

$$P = 15\left\{{\cdot}75 + 4{\cdot}28\left({\cdot}04 - \frac{{\cdot}16}{8}\right)\right\} = 15\{{\cdot}75 + (4{\cdot}28 \times {\cdot}02)\} = 15 \times {\cdot}8356 = 12{\cdot}53 \text{ lbs.}$$

The following table shows the calorific power of coal from widely different sources *:—

Locality.	Nature of Coal.	Calorific Power of Dry Coal free from Ash (in British thermal units).
Toula, Russia,	Lignite,	13,837
Manosque, Basses Alpes,	,,	12,584
,, ,,	,,	13,253
France and Germany,	Brown Coal,	11,340–14,220
England,	Caking Coal,	15,804
,,	,,	16,108
Basin of Donetz, Russia,	,,	15,651
Creusot, France,	,,	17,319
,, ,,	Anthracite,	17,021
Basin of Donetz, Russia,	,,	14,866

* *Coal, its History and Uses,* p. 250.

From recent experiments made on Scotch coals the calorific power and specific gravity are shown in the following table :—

	Specific Gravity.	Calorific Value (B.T.U.)
Ell, ...	1·266	13,464
Main, ...	1·261	13,662
Splint, ...	1·292	13,365
Gas, ...	1·290	13,266
Virgin, ...	1·286	13,464
Kilsyth Haughrigg, . .	1·291	13,563
Bannockburn Main, . .	1·306	14,157
Kilsyth Coking, . . .	1·275	14,058

The calorific value of a pound of fuel in B.T.U. can be calculated from the formula :

$$x = 145C + 620(H - \tfrac{1}{8}O),$$

where C, H, and O represent the percentages of carbon, hydrogen, and oxygen present as determined by analysis.

CHAPTER II.

THE SEARCH FOR COAL.

Boring.—The search for coal in an unknown district is the application of geology to practical uses. In such a search all available means are taken to obtain information, such as the examination of quarries, beds of rivers, and railway cuttings. Even the ploughing of fields has often led to the discovery of the presence of coal when there were no other indications.

An examination carefully carried out in any district will reveal whether the strata belong to the coal-bearing formation or not. The discovery of a few fossils, such as sigillaria or stigmaria, will at once identify the rocks; or an 'outcrop' of coal may be discovered at the surface, but this is not often the case, particularly if the coal-bearing strata have been deeply overlaid by newer formations. In such circumstances resort must be had to boring to decide whether coal be present or not.

Boring is the means adopted to determine the existence of beds of minerals, such as ironstone, coal, and salt, lying below the surface of the earth, and to obtain information respecting their position, thickness, and quality.

The uses of bore-holes vary considerably, and may be stated as follows:—

- To reach a mineral deposit in order to ascertain its nature, depth from the surface, strike, etc.
- To ascertain the nature of the subjacent rocks for engineering purposes, such as railways, canal cuttings, bridges, etc.
- To obtain liquids such as ordinary water, mineral water, brine or petroleum.
- To make absorbent wells in dry and porous strata.
- To obtain gases, such as natural inflammable gas, carbonic acid gas or vapours containing boric acid.
- To drain off water or gas from mine workings.
- To make passages for conveying water to underground fires or power into underground workings.
- To install signal wires or speaking tubes.
- To sink holes for lightning conductors, house-lifts, or piles.
- To introduce cement into unsound foundations to strengthen them, and also into mine workings to dam water.
- To sink mine shafts.

In a field of coal it is usual to put down a series of bore-holes for the purpose of obtaining a correct section of the strata passed through; of finding the depth of a seam or seams from the surface, the thickness, quality, and number of seams, the chemical properties of the coal and the nature of the roof and pavement; and of ascertaining the inclination of the strata, as also the number and size of 'faults' in the field.

In establishing the existence of dykes or faults underground, bore-holes sometimes save time and money which might otherwise be wasted in exploring by means of shafts, particularly when the 'vees' of the fault is nearly vertical or ill-defined, and it is difficult to determine whether it is an up-throw or a down-throw fault.

By their aid the gradient of a road that would intersect the dislocated seam can also be determined.

Methods of Boring.—There are two chief methods of boring, viz., percussive and rotary. These may be again sub-divided thus:—

(1) Percussive Boring	With Rods.	Ordinary method of chipping and removing débris.
		Japanese method of 'plunging' without removing débris.
	With Ropes.	Chinese and other methods with a spring pole.
		Ordinary method employed in American oil districts.
		Special methods, such as Mather & Platt's and others.
(2) Rotary Boring		Boring with augers in soft material.
		Diamond rock-drill boring.

Hydraulic methods are applied to both systems of boring.

The percussive method is largely adopted for shallow bore-holes, or in soft, easily worked rock. It is commonly carried out by means of free-falling tools, which chip or cut the rock into angular fragments. The rotary method grinds the rock into powder, or can be made to cut out a solid core. The commonest method of boring, for depths of 5 fms. or more, is carried out by means of a steel chisel screwed into an iron rod, and suspended from a spring-pole. The tools used for a bore-hole in ordinary strata are chisels, rods, bracehead, augers, and sludgers for extracting the loose material; tools for dressing the sides of the bore-hole and for extracting broken rods or chisels: also keys for screwing and holding the rods, and tubes for lining the holes.

Chisels or Bits.—The form, sharpness, and temper of the cutting tool employed vary according to the rock which has to be cut through. Various chisels are in use: flat or straight-edged for ordinary strata, **V** or diamond-point chisels for hard rock; the **T** chisel for gravel; while others, with cutting edges shaped like an **S** or **Z**, are used for different kinds of work, but these chisels are difficult to sharpen and maintain in good order. For soft ground, such as clay or peat, augers are used. The chisels are 18 in. to 24 in. long, 1 in. to 2 in. diameter, and 2 in. to 3 or 4 in. in breadth of face. They are made of the best steel, and weigh from 3 to $4\frac{1}{2}$ lbs. each. Fig. 6 shows some of the forms used.

Rods.—The rods are made of wood or iron, more commonly the

latter, the best material being selected. They are octagonal, round, or square in section. Ordinary rods are $\frac{3}{4}$ in. to $1\frac{1}{2}$ in. square, $\frac{7}{8}$ in. and 1 in.; they are made in lengths of $1\frac{1}{2}$ ft. to 10 or 12 ft.; the bottom rod is always about 3 ft. long. The usual mode of connecting the rods is by a screw-joint (fig. 7). Iron rods 1 in. square weigh about 10 lbs. per yard. Wooden rods are generally made in 20 to 30 ft. lengths of pitch pine, and not less than $2\frac{1}{2}$ in. square. The sections are joined by ordinary butt, or scarf joints and iron strapping plates.

Bracehead.—For shallow holes boring can be accomplished by the single bracehead, actuated by two or more men, for a distance of 10 or 15 yds.; beyond that depth a double bracehead is used until 20 or 30 yds. is reached, when a spring-pole and windlass will be required. The single bracehead is made with a wooden handle about 3 ft. long and 3 in. diameter at the centre, and tapers at each end. The centre is furnished with an eye made of iron, to which the rods are attached (fig. 8).

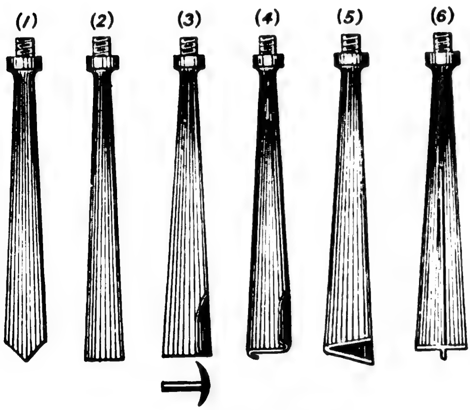


FIG. 6.—Chisels.

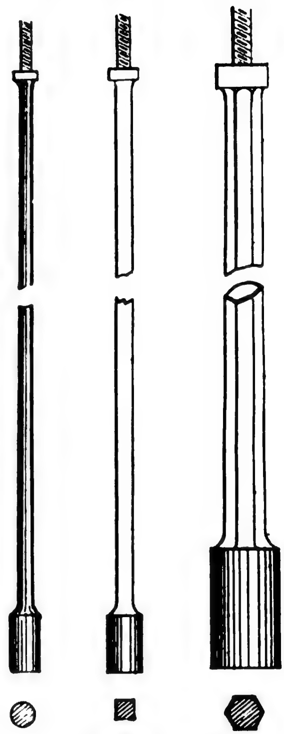


FIG. 7.—Rods.

Sludger.—The sludger is usually a tube 3 to 10 ft. in length, and of a diameter suitable for the bore-hole. It is provided with an ordinary clack or ball valve at the bottom (fig. 9). When it is required to clear the bore-hole, the sludger is lowered, and worked up and down a few times at the bottom in order to fill it with the broken material; it is then drawn to the surface, and the contents carefully examined.

The Bèche is the tool used for extracting broken rods in cases of fracture. It is about 2 ft. long, and hollow for about 16 in. at the lower end, the diameter of the opening at the bottom being about $1\frac{3}{4}$ in. and tapering to $\frac{5}{8}$ in. diameter (fig. 10).

The Brake-staff is a lever of pitch pine, 10 to 14 ft. long, having a fulcrum $1\frac{1}{2}$ ft. to 2 ft. from the end next the rods. At one end is placed an iron hook, a rope being attached to it to enable the men to give it motion (fig. 12).

Preliminary Operations.—When boring is about to be commenced, a platform of wood, 2 or 3 ft. square and 3 in. thick, is laid on the ground, and a hole bored in the centre for the rods to pass through into the bore-hole. Short lengths of rods, 18 in. to 3 ft., are used at the beginning until the hole attains sufficient depth for the ordinary lengths to be used. The hole should at starting be larger in diameter than the deeper portions of the boring are intended to be.

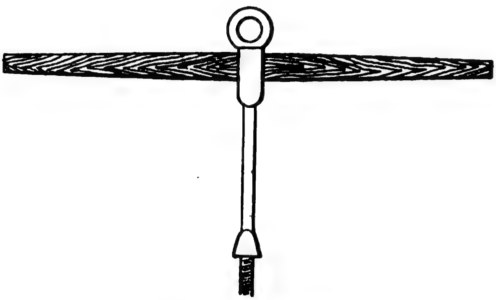


Fig. 8.—Bracehead.

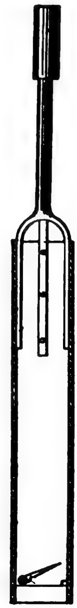


Fig. 9.—Sludger.

If the bore-hole is to be deep it is a common practice to dig a small pit, 6 ft. square and 12 to 15 ft. deep, before boring is begun. This small pit is of great utility, as it gives additional clearance for the withdrawal of the rods, removes the loose material at the surface, and reduces the ultimate cost of boring. During the actual operation of boring by percussion, the rods are raised about 1 to $1\frac{1}{2}$ ft. and allowed to fall suddenly, driving the chisel against the rock. Every time they are raised the master borer gives them a slight turn with the 'tiller' (fig. 11), causing the chisel to deliver a blow in a fresh direction. When the tool has been at work for some time the bottom of the hole gets filled with débris, which has to be removed by the sludger, which is screwed for this purpose to the end of the rods. When the ground is soft, and the grindings fine and sandy, a sand pump, working on

practically the same principle as an ordinary cylinder pump, is used. If the hole has to be deeper than 20 or 30 yds. a boring trestle and frame are erected; the former as a fulcrum for the brake-staff, and the latter in order to raise the rods easily and speedily when required.

In fig. 11 are shown some of the tools used in ordinary boring, but there are many others. In boring it is usual to erect some sort of head-gear to enable the rods to be raised quickly and easily. This head-gear may consist of a triangular frame of three long wood poles, either circular or square, meeting at the top, where they are fastened together by a bolt. The head-gear may be from 20 ft. to 60 ft. in length, the higher the better; but whatever height is adopted, it ought to be a multiple of the lengths of the boring-rods used, so that when the rods are raised, the joint for unscrewing should be just above the top of the bore-hole. Thus, if the rods are in 12 ft. lengths, the boring frame ought to be 24, 36, or 48 ft. high. If the hole is to be a deep one, a steam winch is used, as it raises the rods more speedily, and is more reliable than a hand windlass.

FIG. 10.—Béche.

When the hole reaches a certain depth the rods require to be balanced in some way, as their whole weight, if allowed to fall on the chisel, would damage or break it. To remedy this, the weight of the rods may be transferred to the end of the brake-staff, or a rope can be used instead, and by a suitable arrangement a weight sufficiently heavy to cut the rock is allowed to fall on the chisel. This plan has been adopted in Mather & Platt's system of boring. Some boring engineers prefer wooden rods for boring, for when the hole fills with water the rods are buoyed up, and a great deal of their weight is thus taken off the chisel.

LIST OF TOOLS SHOWN IN FIGURE 11.

1. Spiral worm or miser.
2. Bell screw.
3. Bell box with cleats.
4. Crows-foot.
5. Bell-mouthed shell.
6. Auger shell.
7. Worm or screw auger.
8. Plug drill.
9. Parallel worm auger.
10. Shoe-nose shell.
11. Auger-nose shell.
12, 13. Shell augers.
14. Bow dog.
15. Spring dart.
16. Tillers or levers.
17. Gravel chisel.
18. Clay auger.
19. Reamer.
20, 21. Lengthening pieces.
22. Lifting dog.
23. Nipping fork.
24. Hand dog.
25. Snatch block.
26. Auger cleaner.
27. Holding-up rod.
28. Tie screw-driver.
29. Spring hook.

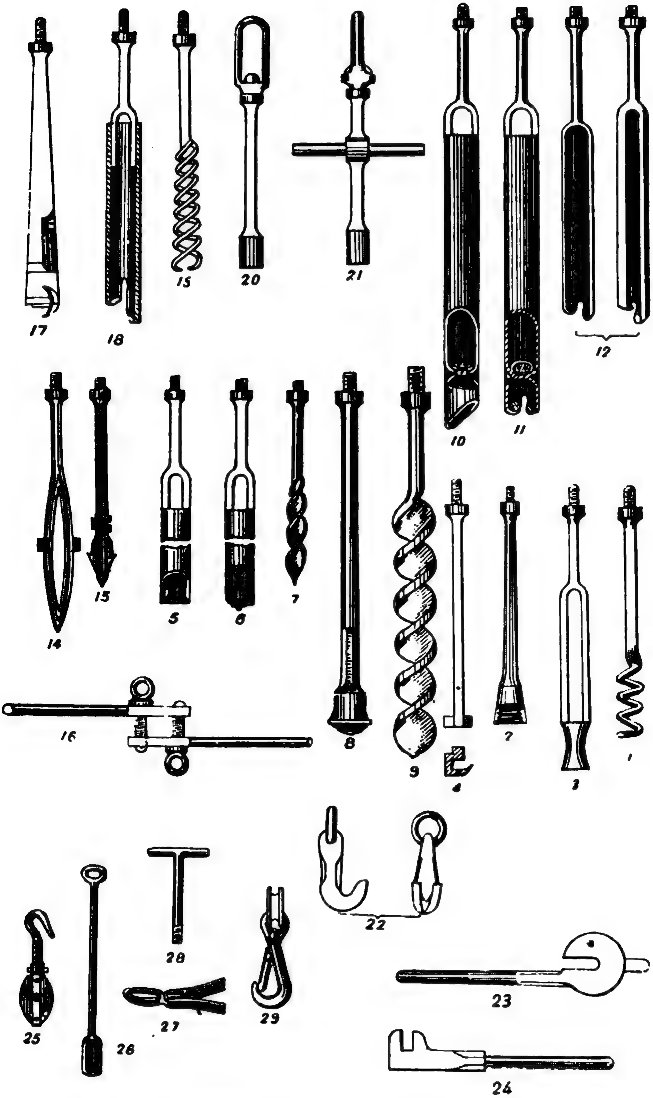


FIG. 11.—Boring Tools. (For list see opposite page.)

Another arrangement to guard against fracturing the rods is the sliding joint, usually fixed 10 to 20 yds. above the chisel, the rods *a a* below it being made extra strong (fig. 13). When the chisel strikes the ground, the upper lengths of rods *b* move over the sliding joint, until the beam to which it is fixed has completed its stroke, when an elastic stop at the upper end helps to deaden the fall, and thus the shock due to the chisel and rods striking the rock simultaneously is avoided.

On the return stroke the collar *c*, suspended from the rods *d* by the fork *f*, catches against the projecting cross-head *e*, and thus lifts the chisel again. If the rods happen to break, the 'Béche,' 'Crows-foot,' or some other grapnel is used to grip and raise them. A simple kind of grapnel is a bell-mouthed tube about 5 ft. long (fig. 14). Near the bottom of the inside of the tube are fixed four steel blades or springs. To extract the broken rods the tube is lowered until it passes over a joint below the fractured rod, the steel blades being pressed outwards when passing this joint, but immediately it is passed they press firmly in on the rod, and the grapnel is then raised, taking the broken rods along with it.

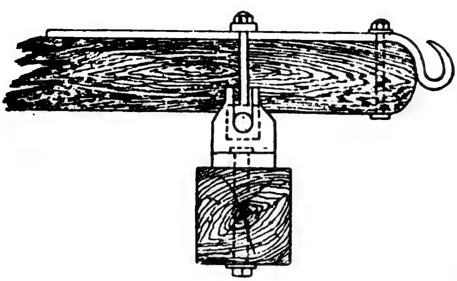


FIG. 12.—Brake-staff.

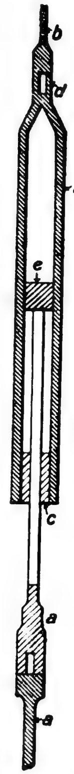


FIG. 13.—Sliding joint.

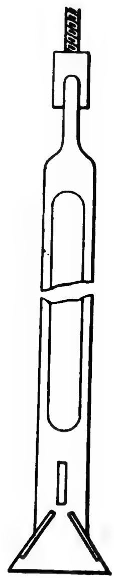


FIG. 14.—Grapnel tube.

A steam or hand windlass to which a rope is attached is usually employed for clearing the hole, and also for raising the rods when the chisels are being changed.

Lining the Bore-hole.—If the sides of the hole are of a soft nature and apt to fall in, tubes ought to be inserted to protect the walls and to allow the rods to work freely. Those used for this purpose are generally made of wrought iron $\frac{1}{8}$ in. to $\frac{5}{8}$ in. thick, in 9 to 12 ft. lengths. They are forced down by repeated blows from a heavy

mallet, but often get bent by such hammering, and it is better to use either screw-jacks or hydraulic rams to force them down. The tubes are made with either screw-joints (fig. 15), or lap-joints with rivets, the former being the more frequently used.

Removing the Lining.—If the tubes can be withdrawn, they are often taken out after the bore-hole is finished, but if they resist removal, they are left standing, as the cost of withdrawing them would, in many cases, be more than the original cost of the tubes. When the friction is not great, Kind's plug or shuttle is used. It consists of an oval ball of wood, of slightly smaller diameter than the tubes. It is fastened to the end of the rods and lowered down; a short, open tube resting on its upper surface is filled with coarse sand. When the desired position has been reached, the short tube is raised by a rope, and the sand spreads over the plug, filling the space between it and the tubes, thus causing sufficient friction for the latter to be withdrawn when the plug is drawn up (fig. 16). If it is difficult to withdraw the tubes, a tool provided with two darts or jaws which work on a spring has to be used. When this tool is being lowered down, the jaws are closed, but as soon as they reach the bottom of the tube they fly outwards, and enable them to be withdrawn (see 15, fig. 11).

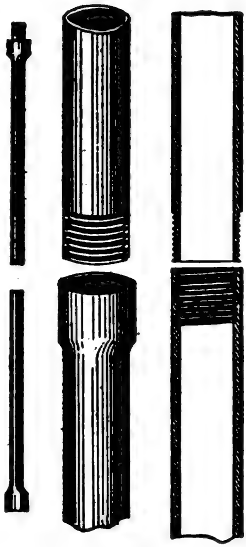


FIG. 15.—Tubes for Lining.

FIG. 16.—Plug.

Speed of Boring.—The speed with which a bore-hole is put down varies according to the nature of the strata passed through and the kind of tools and machinery employed. In ordinary coal-measures the rate of progress may, in the early stages, be as much as 1 ft. per hour; but as the hole gets deeper the speed will be very much lower, owing to the greater amount of time taken up in raising and lowering the rods, and the rate of progress may not exceed 1 or 2 ft. in twenty-four hours. In hard rocks, such as whinstone, the distance bored may not be more than 6 in. per diem.

André gives, as an average speed in different measures, 1 ft. 9 in. per eleven hours.

Cost of Boring.—This will vary greatly according to the nature of the ground passed through and the wages ruling in the district.

In ordinary boring by spring-pole in the coal-measures, the prices ruling in the North of England are usually 7s. 6d. per fathom for the first 5 fms., 15s. for the second 5 fms., 22s. 6d. for the third, and so on, increasing 7s. 6d. per fathom for every additional 5 fms. depth. The prices in Scotland are 4s. to 4s. 6d. per fathom for first 5 fms., 8s. per fathom for second 5 fms., and so on, increasing by 4s. per fathom every step of 5 fms. In other districts the prices are sometimes 5s. 6d. per yard for the first 5 yds., 7s. 6d. for second 5 yds., 9s. 6d. for the third 5 yds., and so on, increasing 2s. per yard for every additional 5 yds. in depth. These prices are generally taken on the basis of bore-holes starting 3 in. diameter.

The cost of bore-hole may be found by the formula,

$$x = \left\{ 2a + (n-1)d \right\} \frac{n}{2},$$

where x = total cost in shillings, a = price of first step, d = increase of price for each additional step, and n = number of steps.

Example.—What would be the cost of a bore-hole 150 fms. deep if the price of boring is 7s. 6d. per fathom for first 5 fms., and increases by 7s. 6d. for every subsequent 5 fms.?

The total number of steps will be $\frac{150}{5} = 30$,

$$\therefore x = \left\{ (2 \times 7{\cdot}5 \times 5) + (30-1)7{\cdot}5 \right\} \frac{30}{2}$$
$$= \{ (75) + (29 \times 7{\cdot}5)15 \} = 4387{\cdot}5\text{s.} = £219\ 7\text{s.}\ 6\text{d.}$$

Japanese Method of Boring.*—This method is chiefly applicable in soft alluvial deposits, which can be bored through at the rate of 50 to 60 ft. per day. The tools used are solid, round, iron rods, connected by fish-joints, and terminating in a pear-shaped solid plunger for soft ground and an obtuse pyramidal point for hard ground. By a rocking lever terminating with a Jizai Kagi, or *kettle-catch* (peculiar to Japan), the rods are pumped up, 6 in. at a time, to a height of from 2 to 15 ft., when they are allowed to fall. Material is but seldom taken out of the hole. During the boring the hole is kept filled with water charged with clay. Subsequently it may be lined with bamboo pipe. The method is cheap and rapid.

Cost of Boring Plant.—The following are some estimates of boring plant costs in chisel boring in ordinary coal-measures.

For a depth of 33 ft., with 1-in. rods and 2-in. boring tools, to include rigger, rope, and shear-legs, ...	£21	0	0
For boring 3-in. and 4-in. holes, 50 ft. deep, with 1-in. rods, 2¾-in. and 3¾-in. tools, with rigger, rope, and shear-legs, .	35	6	0

* *The Miners' Handbook*, Prof. J. Milne, F.R.S., p. 42.

For 3-in. and 4-in. hole, depth 100 ft., with 1-in. rods, 2¾-in. and 3¾-in. tools, with screws for 1¼-in. bottom rods, to include rigger, rope, and shear-legs, ...	£44 15 0
For holes 150 ft. deep, 1¼-in. rods, 3¾-in. and 4¾-in. tools, with rigger, rope, and shear-legs, ...	57 5 0
For holes 200 ft. deep, 1¼-in. rods, 3¾-in., 4¾-in., and 5¾-in. tools for boring 4-in., 5-in., and 6-in. holes, with screws for 1½-in. bottom rods, and with rigger, etc., ...	70 0 0
For holes 300 ft. deep, with 4¼-in., 5¼-in., and 6¼-in. tools, to include rigger, rope, etc., and geared windlass, . .	120 0 0
For holes 500 ft. deep, same as No. 6, with £35 extra for rods,	155 0 0
For holes 800 to 1000 ft. deep, with boring tools, etc., and 800 ft. 1½-in. rods, ...	195 0 0

Difficulties and Accidents in Boring.—These, which are common to all methods of boring, may be briefly summed up as follows:—

Brittleness induced in rods by repeated blows and vibration.
Rods, chisels, bolts, and other tools, by carelessness, fall into the hole, and their extraction gives great trouble.
If the rods are not kept rotated, projections will be left on the sides of the hole, which will lose its circular form and cause the rods to get fixed or broken.
The hole may deviate from the vertical, and so cause the rods to get jammed.
The sides of the hole may fall in and cause much labour and expense in clearing again.
By fissures or change of strata the hole may become narrower.

Mather & Platt's System.—In this system of boring, the ordinary rods are dispensed with and a rope substituted for them. The rope *a* passes over a pulley (*b*, fig. 17), and is guided by another pulley *c* to a drum, which is worked by a horizontal cylinder. Between the drum and the pulley a clamp *d* is attached to the boring frame. The rope between the top pulley *b* and the drum is fixed by this clamp during the operation of boring. A vertical cylinder *e* connected directly to the pulley actuates the boring rope to which the boring tool is attached. Steam is admitted into this vertical cylinder *e* on the bottom side of the piston, and the wheel *b*, which works in a sliding frame, is raised along with the rope and boring tool. When the piston has completed its upward stroke, a valve *f* at the bottom of the cylinder is opened, allowing the steam to escape, and at the same time the boring tool falls rapidly, giving a smart blow at the bottom of the hole. The length of the stroke can be varied by self-acting tappets at the bottom of the cylinder. As the wheel *b* can be rotated as well as raised vertically, the boring tool will be raised 2 ft. for every foot the piston is raised, and hence on the downward stroke it will fall with twice the speed of the piston. The horizontal engine and drum attached are only for raising and lowering the boring tool and the sand pump or sludger. The rope is generally flat and of hemp, 4½ in. broad by ½ in. thick.

In this system of boring, the great difficulty at first was the

turning of the boring tool at each stroke. For this purpose an ingenious arrangement was adopted.

The sliding collar (*g*, fig. 18) has a series of teeth top and bottom. On raising the rope on the up stroke, this collar *g* fits into another fixed collar *f* at the top, the lower edge of which has a corresponding set of teeth. When the tool falls and strikes the blow at the bottom of the hole, the sliding collar *g* descends and fits into another collar *e*, having teeth which are set half a tooth in advance of those in the sliding collar *g*. Thus, when the tool falls, the inclined surface of the lower teeth in the collar *g* strikes the point of the teeth in *e*, and finally fits into them, thereby giving the flat rope a turn of half a tooth, and on the tool being raised it is twisted another half tooth, when the sliding collar comes again in contact with the teeth in *f*. The rope therefore receives altogether a twist to the

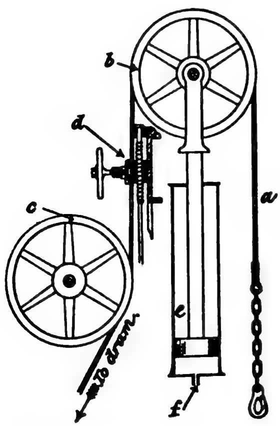


FIG. 17.—Mather and Platt System.

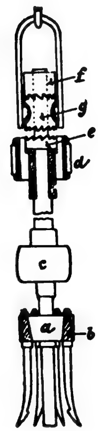


FIG. 18.—Sliding collar.

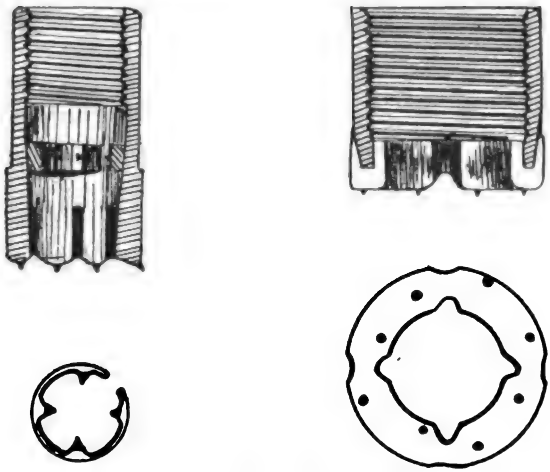
extent of one tooth, and in untwisting it must turn the tool a like amount, causing it to strike in a fresh place. This constant change goes on automatically, and secures the efficient cutting of the rock. The boring tool can be lowered at the rate of 500 ft. per minute, and can deliver about twenty-four blows per minute, the rate of boring being 2 to 4 ft. per diem in deep bore-holes. The cost of boring seems, however, to be more expensive than other systems, a bore-hole at Middlesbrough, 1200 ft. deep, costing no less than £8, 6s. 8d. per foot.*

Messrs Mather & Platt have recently introduced a machine similar to the method employed at Pennsylvania oil wells, *i.e.* by means of a circular rope, attached to the end of a large oscillating

* *Trans. N. Eng. Min. and Mech. Engs.*, vol. xxx. p. 88 ; and Foster's *Ore and Stone Mining*, sixth edition, pp. 156 and 324.

beam, which is actuated by a large belt wheel from an engine, by a connecting rod and crank.*

Diamond or Rotary Boring.—In this system of boring, a solid core of the strata passed through is cut out by means of diamonds set in a cylindrical crown, which is fastened to the foot of the rods, and is furnished with grooves for the circulation of water. At the foot of the boring rods there is a cup 3 ft. long and 4½ in. inside diameter, by which any sediment, too heavy for the water to carry away, is caught. This cup and the rods are attached by two plugs to a core tube 12 ft. long; at its lower end there is a nipple for connecting it with the crown (figs. 19, 20). At the junction of the two plugs there is a loose steel ring, inside which the core tube and



Figs. 19 and 20.—Diamond drills.

crown revolve, and which serves to prevent the tube from getting worn by friction with the sides of the bore-hole. Inside the crown there is a spring trap for extracting the cores. Its sides are parallel throughout its inside diameter, but bevelled on the outside to allow of its rising and falling. When the crown is revolving the spring trap is 'up,' in order to allow the core tube to rotate freely, but when a core is about to be drawn the trap is jammed down to permit the core to be broken off as required. The boring rods are tubes about 1¾ in. diameter, made of mild steel, in lengths of 12 ft., and weigh about 50 lbs. each. A constant stream of water is forced under pressure through these rods at the rate of 12 to 16 gallons per minute. On reaching the bottom this water circulates outside the rods, returning ultimately to the surface, and carrying with it

* *Trans. N. Eng. Inst. Min. and Mech. Engs.*, vol. xxx. p. 88.

any sediment made by the cutting of the core. The rods are rotated by gearing driven by a pair of small vertical engines with cylinders 6 in. diameter and 10 in. stroke, the revolutions of the rods being at the rate of 100 to 200 per minute.

Figs. 21 and 22 show the method of working. At the beginning of the bore the rods are weighted (fig. 21) to give sufficient pressure to the diamonds, the weight being from 15 to 20 cwts. in soft strata

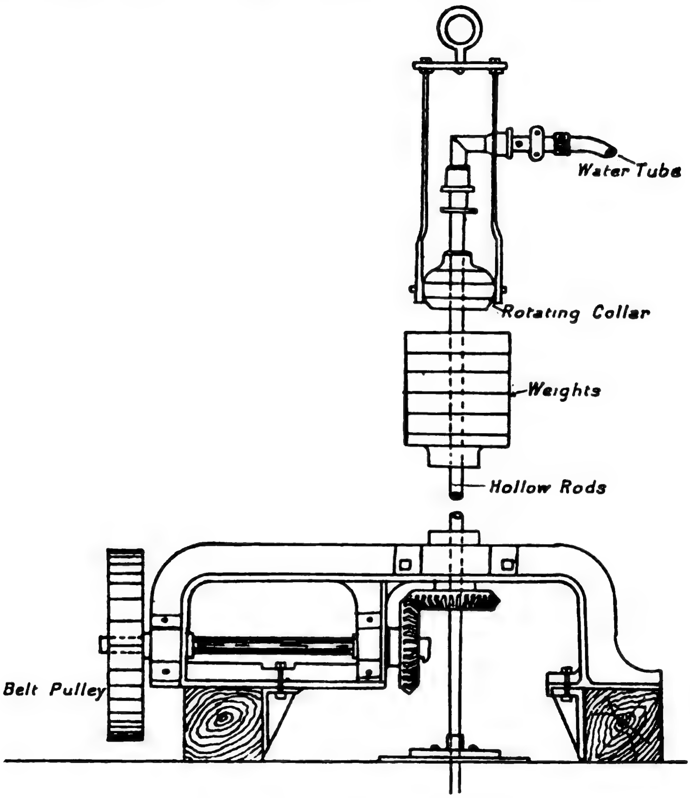


FIG. 21.—Boring frame. *Elevation.*

and from 20 to 30 cwts. when hard strata are being cut. As the depth of the hole increases, these weights are gradually taken off, the weight of the rods alone being sufficient to cause the required pressure, while, if the hole be a deep one, part of the weight may require to be counterpoised by others. The diamonds used for boring are what are known as 'Boart,' or fractured diamonds, costing about 12s. a carat. In a crown $5\frac{1}{4}$ in. diameter, sixteen to eighteen

stones would be used, costing from £35 to £40. The whole crown, including diamonds, would cost about £80.

Speed of Diamond Boring.—In good firm strata, and with a speed of 100 revolutions per minute, 30 to 40 ft. may be bored through in nine hours; but either in very hard rock or very soft strata the rate would be sensibly reduced, particularly if the hole becomes deep. A bore-hole was put down at Newton in Lanarkshire by Messrs Thomson Brothers, Dunfermline, the depth of which was 149 fms. 3 ft. 10 in. It was commenced on 19th December 1889, and finished on 25th March 1890, the total number of days worked being sixty-seven, giving an average rate of cutting of 13·4 ft. per day of nine hours. To attend to the operations three men were required, a leader and two assistants.*

Cost of Diamond Boring.—In the North of England the cost of diamond boring is 8s. per foot for the first 100 ft., increasing 8s. per

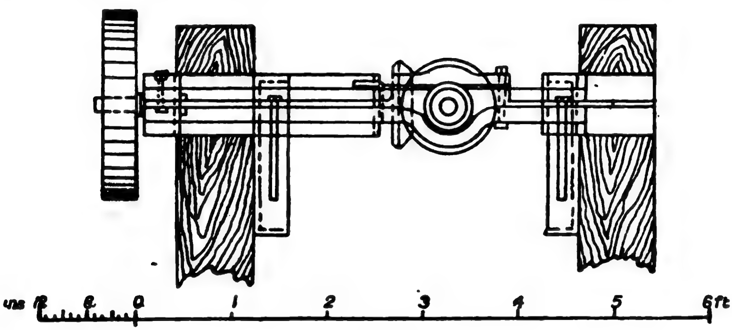


FIG. 22.—*Plan.*

foot for every additional 100 ft. In other districts, bore-holes not exceeding 1000 ft. are charged £1 per foot, 1000 to 1500 ft. deep £1, 10s. per foot, 1500 to 2000 ft. deep £2, 10s. per foot. In Scotland the cost of diamond boring is £3 per fathom for depths up to 150 fms., and from 150 to 250 fms. deep £5 per fathom.

For these rates the boring firm usually supply all labour, tools, and machinery required, those for whom the bore is being made paying railway costs and cartage to the site of the bore, and also providing the necessary lining tubes, fuel supply for engine, and water supply required for the boring operations.

Hand-power Boring Machine.—For shallow holes in ordinary strata, such as are generally met with in the coal-measures, hand diamond drills of simple construction, and readily transported from place to place, can be obtained at prices varying from £160 to £200, according to the prevailing prices of diamonds.

Advantages of the Diamond System.—The advantages claimed for

* *Trans. Min. Inst. Scot.*, 1891, p. 156.

this system are that it is more expeditious than methods of boring by a chisel, and that, cores of the strata being obtainable, a correct section of the rocks passed through can be ascertained and the inclination of strata seen, while for deep holes it is cheaper than the chisel method. In very soft strata the cores got are not, however, very satisfactory.

Davis Calyx Drill.—This system of boring, on the rotary principle, closely resembles diamond drilling.* The boring bit, or cutter, consists of a cylindrical metal shell, the lower end of which has been formed, by a process of gulleting, into a series of long sharp teeth. These teeth are set in and out alternately. Those having an outward set are used to drill the hole just large enough to allow the apparatus to descend freely, and the teeth having the inward set dress down the core to such a diameter as to allow the body of the cutter to pass freely over it without binding. The front face of each tooth is perpendicular at the base to the rock to be operated on; while the back of the tooth rises from the same line at an angle of about 60°. Immediately above the cutter is the core-barrel, which is connected to the boring rods by means of a reducing plug, which also serves to close the lower end of the calyx. The calyx is simply a long tube or series of connected tubes, located above the core-barrel, to which it is equal in diameter. The lower drill rods work through the centre of this calyx, there being an annular space between the two, and at the upper end the calyx is kept concentric with the drill rods by means of a centering device.

Mode of Operation.—When drilling is being carried on, a continuous stream of water is pumped down the drill rods, in the same way as in diamond drilling, the drill at the same time being slowly rotated and forced downward. The rods have to be twisted considerably before they accumulate sufficient energy to overcome the 'bite' of the teeth into the rock, but the moment the surface strain exceeds the resistance below, it begins to grip into the strata by a series of motions similar to that of a stonemason's hammer and chisel action.

The débris which is produced by this action in the formation of the core is carried up by the stream of water to the top of the calyx, where, owing to the reduction in velocity of the water flow, they slowly fall back into the annular space between the drill rods and the inside of the calyx tube. It is claimed for this system that a longer core can be produced and more accurate results obtained than with the diamond drill, and that the rate of boring is greater.

Other Objects of Boring.—Boring has often to be carried out underground when old workings, supposed to contain water or gas, are being approached. In these circumstances, a pair of narrow drifts 6 to 9 ft. wide are driven in advance of the working faces at the nearest point to where the waste is to be tapped. Bore-holes are kept in advance of the face for a distance of not less than 15 ft., and

* *Trans. Inst. Min. Engs.*, vol. xv. pp. 363–366.

also sufficient flank holes set at an angle of about 45° with the drift. If the coal and roof are 'tender,' and the pressure of the water great, the advance holes will require to extend a good deal further than 15 ft. Whatever length is decided on ought to be adhered to throughout, and the greatest care should be taken, as old plans cannot be implicitly relied on. The holes are usually bored with light iron rods $\frac{3}{8}$ in. to $\frac{1}{2}$ in. square, and in 6-ft. lengths, the drill cutting a hole about $1\frac{1}{2}$ in. diameter, the holes being sloped a little towards the roof.

If the old workings take the form of an irregular boundary of 'stoop and room,' there is always a danger of an open 'drift' being passed by the exploring road at a short distance from it, so that if the flank holes are not sufficiently close to catch the old roads, the water may break through on the side of the exploring drift, possibly at some distance back from the face. After the waste is tapped, it may be necessary to regulate the escape of water to suit the capacity of the pumping plant, otherwise the pit may become flooded. This is often done by the insertion of a tube in the last length of the hole, into which are fitted wooden plugs, 5 to 8 ft. long, tapered 1 in. to $2\frac{1}{2}$ in., having a hole bored through their centre to allow just as much water to escape as the pumping arrangements can adequately deal with. The tube may also have a tap fixed to it, when the water can be drawn off as required. If fire-damp is suspected, only safety-lamps should be used, and spare lamps kept ready lighted at some distance in the rear. Precautions should also be taken to provide against other gases escaping in dangerous quantities.

In tapping wastes by the ordinary method of drilling holes, over, on an average, 20 ft. in length, there is always much difficulty experienced in getting the tools to clear themselves; indeed, they often get choked up altogether with the loose débris, and occasion much trouble in withdrawal. To remedy this difficulty, boring machines specially adapted for the purpose are sometimes used. Such a machine is shown in fig. 23.

The machine consists of a cylinder *c* $1\frac{1}{2}$ in. diameter, furnished with packing glands through which a spindle *s*, connected to the boring rods, is worked.* Fixed on the frame is a pump-chest D, which is connected to the cylinder *c* by means of an india-rubber pipe E $\frac{3}{4}$ in. diameter. In this pump-chest are two small plunger pumps, 1 in. diameter and 1 in. stroke. These pumps are worked off two cranks on the spindle *s*, and are supplied with water by a pipe leading to a cistern. On the outer end of the spindle is a handle for working the machine. The whole apparatus is placed on a bogie running on ordinary rails. The machine is kept moving forward, while the drilling proceeds by means of a chain fixed to a barrel *b* with a ratchet wheel; the chain passes round two other pulleys *p p'* fixed to a prop *h*, and a weight W is hung at the other end of the

* *Trans. Min. Inst. Scot.*, vol. xiii. p. 82.

chain. The rods, which are hollow, are $\frac{7}{8}$ in. diameter outside with $\frac{1}{2}$ in. diameter opening, and are made in 6-ft. lengths. They are connected to each other by the ordinary method of screwing. The drill is also hollow and $1\frac{1}{2}$ in. outside diameter, and is of the ordinary description used for drilling holes, except that $1\frac{1}{2}$ in. from the point is an opening to allow the water to escape. In applying the machine, the handle is rotated, and this in turn works the cranks attached to the two force pumps which force the water into the cylinder *c*, and thence into the hollow rods, which carry it forward to the drill point, where it is discharged, and flows back through the hole, carrying the débris along with it. Holes have been bored with this machine for distances of 150 to 170 ft., the rate of cutting being on an average 30 yds. per shift of eight hours, employing two men. The machine can be used either in ordinary strata or in coal.

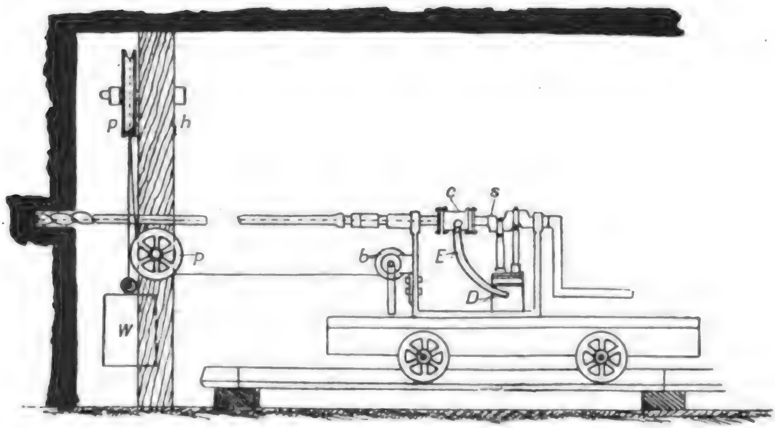


Fig. 23.—Boring machine.

Old wastes may also be tapped by diamond drills, used in much the same way as when employed at the surface. At the Carron Company's Collieries at Bishopbriggs, near Glasgow, bore-holes were drilled in this manner into old wastes containing water, the distance being about 120 ft. The power was derived from a small Priestman oil-engine developing $3\frac{1}{2}$ horse-power, with a consumption of 3 gallons of oil in eight hours. The speed of the engine was 150 to 250 revolutions per minute, while that of the drill was 75 to 125 revolutions per minute. The ignition in the cylinder of the engine was obtained by electric spark from a bichromate battery, which served for about 500 hours.

The speed of boring in the different strata per shift of eight hours with two men was, in sandstone, 30 ft., shale, 18 ft., ironstone, 15 ft., and coal, 47 ft. Better results could have been obtained, but the

position where the drilling was being carried on being confined, short lengths of rods, 18 in., 36 in., and 54 in. long, had to be used at first, which necessitated frequent changing.

Other machines used for boring are the Burnside Safety Drilling Machine,* designed for boring long holes, for tapping flooded workings, and a machine made by Mr Robert Martin, which is really an adaptation of an ordinary miner's ratchet boring machine, and which can be worked by hand. It has been found most useful in the highly inclined workings of Niddrie Colliery, holes from 50 to 70 ft. in length being bored with no greater difficulty than attends the boring of shot holes.

Surveying Bore-holes.—Bore-holes are very apt to depart from the vertical, and may thereby give misleading results; so that it often becomes necessary to ascertain the amount of deviation. This can be measured by the aid of a clinograph or clinostat, an instrument which was first invented and used by Mr E. F. Macgeorge in the Colony of Victoria, Australia. It consists of two glass bulbs, the upper one carrying a plummet and the lower one a magnetic needle, both bulbs being filled with liquid gelatine. The needle is so arranged that it can swing freely without touching the sides of the glass bulb, and so set itself in the magnetic meridian. The small glass cylinder, terminating in the bulb at the top, is inserted through an air-tight cork and a brass capsule at the upper end. This upper bulb contains a delicate plummet of glass, with a diminutive hollow float at the top and a solid ball at the bottom, which is prevented from falling out by a delicate grating. It is carefully adjusted to the specific gravity of the solidifying fluid used, and is so arranged that it will assume a vertical position whenever it is free to move.†

To use the instrument, six of the bulbs are placed in a bath of warm water, heated nearly to boiling point, and a brass cylinder is also heated by filling it several times with hot water. The clinostats are then put into this cylinder one after the other, and lowered into the bore-hole, where they are allowed to remain for two or three hours. By this time the gelatine will have 'set,' fixing the needle in the direction it had assumed prior to the solidification. The brass cylinder is then withdrawn, and the clinostats are examined one by one in an instrument specially designed for the purpose of ascertaining the deviation. Bore-holes may be brought back to the plumb, if deviated, by forcing an india-rubber washer down to a depth of 20 yds. or so beyond the point of deviation, and then running in liquid cement to some feet above where the hole has deflected. The cement is allowed to properly harden, when boring may again be commenced in the right direction.

* *Trans. Inst. Min. Engs.*, vol. xxiii. pp. 75–78.

† For full description of this instrument, see *Mine Surveying*, by Bennett H. Brough, eleventh edition, p. 323.

CHAPTER III.

SINKING.

Preliminary Considerations.—After a coal-field has been sufficiently proved by boring, and the seams have been found to be of good quality and of sufficient thickness to be payable, the sinking of shafts to 'win' the coal has next to be considered. Sinking operations may be divided into two stages: viz., (1) sinking through the surface deposit; (2) sinking through the regular strata.

The surface deposit is often thin and firm and easily sunk through; at other times, however, it is the most difficult part of the work. This is particularly the case when a bed of running sand or mud, a thick bed of mud and boulders, or a bed of peat moss, is met with, such formations presenting considerable difficulties to shaft sinking.

Before starting to sink, several important points have to be disposed of, among which may be mentioned the following:—

- The extent of the field or royalty to be worked.
- The number and thicknesses of seams to be worked.
- The output to be produced per day to yield a profit on the capital invested.
- The quantity of water likely to be met with.
- The number of shafts required, and what their size must be.
- What their positions should be as regards markets, railway communication, etc.
- What faults, dykes, or dislocations exist.

Size and number of Shafts.—The size and number of shafts required will largely depend on the first two heads: the extent of the field leased, and the number and thickness of seams to be worked. Two shafts are the minimum required under the Coal Mines Regulation Act, as a single shaft is only allowed under very exceptional circumstances (see rules 37, 57, 58). The size of the shaft will largely depend on the possible output per day, and on the number of years the coal-field is leased for, or can be exhausted in; points which may be decided approximately as follows:—

$$(1)\ \text{Tons required to be raised per year} = \frac{\text{Total tons of royalty}}{\text{Number of years in lease}}.$$

$$(2)\ \text{Tons required to be raised per day} = \frac{\text{Tons required to be raised per year}}{\text{Number of working days per year}}.$$

In England leases are generally for from thirty to ninety-nine years. The number of years for which leases are usually granted in Scotland is twenty, twenty-five, or thirty.

The size of the shaft will also largely depend on the depth of the seams from the surface, and the amount of water to be pumped. For an output of 300 tons or less per day, with depth not exceeding 240 yds. and pumps 12 in. diameter, a rectangular shaft 14 ft. × 6 ft. inside the lining would be quite large enough. If circular, a shaft 10 ft. in diameter would suffice, under such conditions. For outputs above 300 tons and up to 1000 tons per day, a shaft to hold two double cages would be necessary, and this would require increased shaft space. A rectangular shaft 23 ft. 6 in. × 6 ft. 6 in. inside the lining, or a circular shaft 16 or 17 ft. diameter, would, approximately, meet the requirements.

Site of Shaft.—In choosing a site for shafts, if the surface conditions are suitable, it is generally best, in the case of a large royalty, to sink as near to the centre of the field as possible, for if this site be chosen, the pits will not be so deep—except where the royalty takes the form of a basin with the deepest part in the centre—as if sunk at the extreme dip; and the length each ton of coal has to be hauled will also be less.

If the amount of water given off in the workings is very large the shafts should be sunk so as to have a larger area to the rise than to the dip, as pumping large quantities of water from dip workings is very expensive. If, on the other hand, the workings be dry, the opposite plan is adopted, as haulage, etc., is usually cheaper from the dip than from the rise, the latter usually necessitating long inclines, which become, in time, difficult and expensive to work.

The long side of the shaft is usually sunk in the line of the dip of the seam, so as to get the main roads from the pit bottom set off to level course. If the shafts are not sunk in the line of the dip, then to get the road's level course at the pit bottom, the pavement (floor) would require to be cut on one side, while on the other side it would require to be banked up.

As a rule it is best to sink the shafts to suit the underground workings, and arrange the surface accordingly. The site should also be chosen advantageously in regard to the transit of coal by road, river, or rail, and close to a supply of good water for boilers, etc. It is a very common custom to sink the shafts close together, so as to concentrate the banking and coal-cleaning arrangements. The landowner sometimes stipulates in leases where they are to be sunk.

Form of Shafts.—Shafts may be either rectangular, circular, or elliptical. The rectangular form is almost exclusively used in Scotland and very commonly in America, while in metalliferous mining it is nearly always adopted. Circular shafts are employed in England and Wales for coal-mining, and also on the Continent. In France, and occasionally in Wales, elliptical shafts are used, and in

some parts of Fifeshire, square shafts; but this form is not to be recommended, especially in the case of large shafts. Each of these forms has its own advantages, the circular being the best adapted to resist heavy pressure, and therefore suitable for deep shafts. This form is also best suited for ventilating purposes, as there is always a certain amount of space unoccupied by the cages. The rectangular form of shaft is more economical to sink, easier lined and secured, and the space can be better utilised for winding, pumping, and other arrangements, while less material requires to be excavated.

The author of a paper, read before the Institution of Mining Engineers, in discussing the merits of the different forms of shafts, says, "that in deciding the form of shaft, he fails to understand why in some districts oblong shafts are sunk in preference to circular ones, unless the object is to take out as little ground as possible. It seems desirable, in order to wind a given quantity of coal per day, that there should be the same area (over and above the space occupied by the cage at meetings) in the one shaft as in the other to admit of adequate ventilation. This consideration is in fact so important that in many instances it is deemed desirable to sink a staple pit for a height of some 120 feet, having a holing into the shaft above and below meeting places, so that the area for ventilation may not be diminished when the cages are passing each other. The timbering in an oblong shaft will not last so long as the brickwork in a circular one, and seeing that the shaft is the one entrance into the mine through which all the men must pass in going and coming from their work, it is desirable that it should be made as safe as possible by being securely lined throughout."

Commencing Operations.—The position and size of the shaft having been pegged off, the surface soil is removed, the sides being supported with a temporary lining, if required, until the 'rock-head' is reached, when a perfectly level bed is prepared on which to lay the first walling crib. If the shaft has to be lined with brick, the walling may be started forthwith, but very often the sinking is carried on to a considerable depth by the aid of temporary lining, and walled up afterwards.

To allow for temporary lining and walling, the diameter of the pit will require to be a good deal larger at starting than the ultimate finished size; thus a pit intended to be 15 ft. diameter inside the walling should be begun with a diameter of 17 or 17½ ft. at least.

The curbs that are used are generally made of oak, cut into segments to fit the circumference of the shaft, when the latter is circular. They vary in size, being commonly 6 in. × 4 in. or 9 in. × 4 in., while sometimes they are made 6 in. square. The segments are dressed and fitted together at the surface, and are then sent down the pit and jointed together by cleats. Figs. 24 and 25 show the construction of some of these oak curbs.

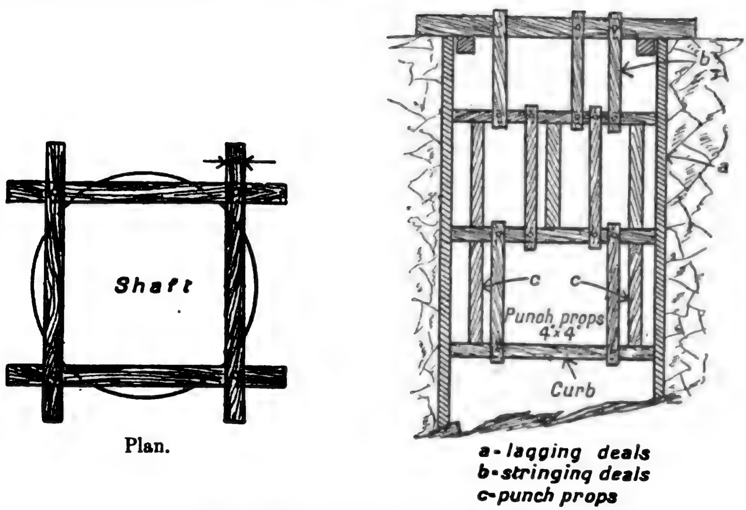
Where the shaft has to be supported with temporary lining, a

square frame made of four strong oak beams, 12 in. to 18 in. square, and firmly bolted together, is laid across the top of the pit, and to this the wood lining is secured. Figs. 26 and 27 show how this temporary lining is fixed in the shaft. When a curb is placed in position, 'backing deals' or 'lagging' of white pine boards, 6 to 9 ft. long by 9 in. to 12 in. broad, and $1\frac{1}{2}$ in. or 2 in. thick, are fitted in

Plan. Elevation.

FIGS. 24 and 25.—Oak curb.

all round between the sides of the shaft and the curb. When these are fixed, other curbs are put in, about every 6 ft., until the surface is reached, and are kept in position by 'punch props' inserted between them, all round the shaft; and finally they are secured to the cross-beams at the surface by means of 'stringing deals,' which are also carried down from curb to curb as the work proceeds (fig. 35). If



Plan.

FIGS. 26 and 27.—Temporary lining.

this temporary lining continues to any great depth, its weight may be transferred to other beams fixed in the shaft.

It has now become the custom to discard the use of wooden curbs, and to substitute iron skeleton rings in order to keep the temporary lining in the shaft. Behind these rings are placed the lagging, or backing deals, which consist of planking 2 to 3 in. thick. The iron rings are easier to handle and put together than wooden curbs.

When the walling of masonry is about to be commenced, the shaft is 'laid back' for 2 ft. or 3 ft., and an even bed prepared for a walling crib; the crib being carefully laid, the walling proceeds, and is carried up for a distance of 10 to 15 yds., when another crib is placed in position. These cribs are now made of cast iron, cast in segments to suit the circumference of the shaft. When the strata are hard and strong, and a section of brickwork is to be put in, no crib need be used; the walling is simply started direct off the rock, and carried up in the usual way. A satisfactory method of proceeding is to carry sinking on for part of the week, and walling operations for the rest of the time. The method of sinking and walling simultaneously is described later. In starting to sink below the walling, the shaft is cut narrow at first, and gradually widened out to its proper width, so as to leave a ledge of rock to support the walling

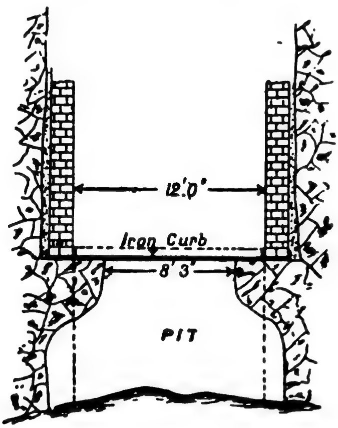


FIG. 28.—Diagram of sinking.

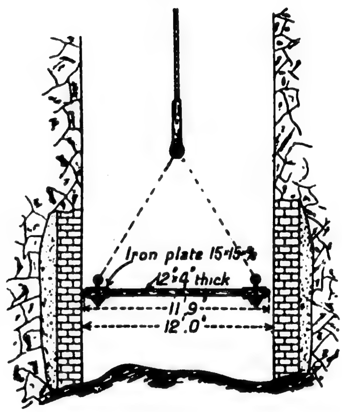


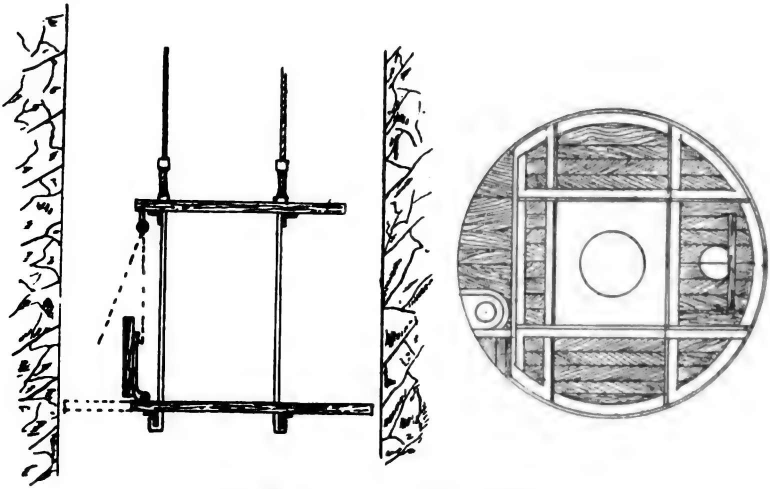
FIG. 29.—Hanging scaffold.

above (see fig. 28). When, at a later stage, the walling is being built up from below, this ledge is not removed all at once, but it is taken out in sections around the circumference, these sections or arcs being then built up to the walling above until the junction is made good all round. The masonry is not usually built back close to the strata, but a small space is left, which is filled with fine ashes or other porous material, so that it helps to relieve the pressure arising from the sides on the brickwork. The walling is carried on by means of a 'hanging scaffold' or 'walling cradle,' made up of 3-in. or 4-in. planking, bolted together and made to fit the curve of the shaft. It is usually made with 1-in. or 2-in. clearance all round, and guides the workman in the building of the brickwork. The 'cradle' is secured to the winding rope by four chains, the latter being fastened to the scaffold by eye-bolts (see fig. 29). The scaffold

should be made to fold up at the centre, so that when a section of walling is completed it can be folded and slung to the sides of the shaft, instead of being raised and lowered each time.

Sometimes the walling is carried out by the help of iron sections, fixed all round the outside circumference of the walling scaffold. The height of these sections is from 2 ft. to 3 ft. 3 in. The brickwork is placed around and in contact with it. When a circular tier of walling is thus completed, the iron section is raised and another tier of masonry placed in position. In this way the time usually spent in measuring the diameter and ascertaining whether the masonry is vertical is saved.

When the space between the walling and the strata is to be filled



FIGS. 30 and 31.—Galloway's scaffold.

in with concrete or cement, the larger size of iron circle is used, so that more may be left below, as a support, until the cement sets.

Sinking and Walling Simultaneously.—The work of sinking has in ordinary cases to be suspended while walling is being performed. In order to obviate this, a scaffold invented by Mr William Galloway is used, consisting of a wooden floor, fixed to an angle iron frame (see figs. 30 and 31). There are two platforms, and access between them is effected by means of an iron ladder.

On one side of the bottom platform is placed a hinged door, which can be raised, to admit of material being raised or lowered. The scaffold is steadied by two guide ropes, and as it occupies nearly the full area of the shaft inside the walling, it saves the time usually taken in measuring the diameter every few yards. At Newbattle Colliery a scaffold, constructed on this principle, was

used for putting in the brick lining a shaft 1650 ft. deep, 20 ft. diameter, walled from top to bottom. The construction of the scaffold will be understood from figs. 32 and 33. The cradle as here used consisted of a working floor *a* and a projecting roof *b*. Between these, ample height is left for men to work. The centre of the cradle contains an opening providing space for two buckets or 'kibbles' to pass each other. This opening is fenced by a circle of sheet iron 6½ ft. high and ¼ in. thick, bolted to the six upright angle bars and hangers.*

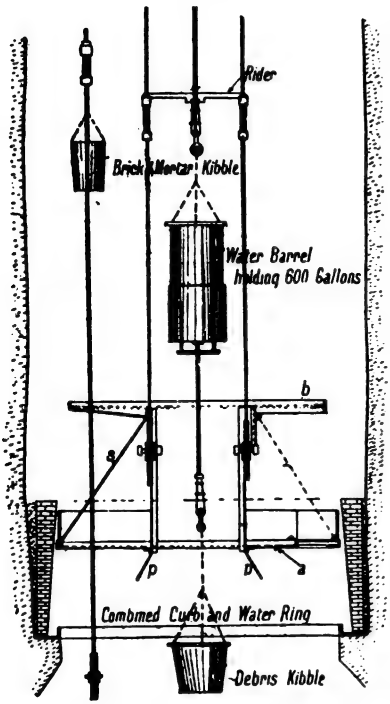


FIG. 32.—Section of scaffold used at Newbattle Colliery.

The floor stage was formed by a 4 in. × 4 in. × ½ in. mild steel angle-bar, bent to the curvature of the shaft, supported on and fastened to the bottom frame or flooring, which was constructed of 5 in. × 5 in. × ⅝ in. steel angle-bars. A door or hatchway was hinged to the floor and raised or lowered by block and tackle fastened by shackles to the door or framing of the scaffold. The door was recessed to allow of its closing the air-boxes and pipes. Hatchway doors were also left on the other side of the cradle to permit of the brick kibble being lowered through if required. The roof of the scaffold was made of 5 in. × 5 in. × ⅝ in. steel angle-irons covered with ⅜-in. sheet-iron plates. Where the suspending wheels carried the whole of the cradle, the framing of the roof was constructed of double 5 in. × 5 in. × ⅝ in. steel angle-bars riveted together.

The floor and roof of the scaffold were connected together by four corner steel angle-bars 5 in. × 5 in. × ¾ in. bolted to the top and bottom framing. Four stays *s s* were placed to the four upright bars for the purpose of supporting and stiffening the roof framing. Two tension-rods 1½ in. diameter were secured to the double angle-bars and to the outer ring of the floor for the purpose of strengthening and carrying the floor.

* *Trans. Fed. Inst. Min. Engs.*, vol. viii. p. 118.

A sheet-iron ring, $\frac{3}{16}$ in. thick, was bolted to the circular framing, and formed a fence round the scaffold 18 in. high. The difference between the diameter of the pit and the upper side of this ring was about $1\frac{1}{2}$ in., *i.e.* an opening of $\frac{3}{4}$ in. existed between the brickwork and the fence. Fending plates *p p* were bolted to the bottom of the scaffold to guide the bucket into the opening of the floor. The weight of the scaffold was about $8\frac{1}{2}$ tons, but with a full working load became about 20 tons. The scaffold was suspended in the shaft by four ropes in double purchase, the ropes being 5 inches in circumference and made of Bessemer steel wire. One end of these ropes was made fast at the surface by attachment to a heavy screw which served the purpose of adjustment in the event of the ropes riding unevenly on the drums. The other end passed round the wheel on the cradle and over another wheel on the winding frame and thence to the crab drum. Four drums were thus required on the crab engine. These ropes between the scaffold and the pulley on the frame also served for guide ropes for a rider to run on, to guide the kettle in the shaft. In addition to these, another guide rope, made fast at the top like the others and looped down the shaft, was provided. Upon this was hung a pulley supporting a water tank and scaffold with two pulsometers, and the suction and delivery pipes. The double rope was brought through the cradle and the loose end led to one of the two drums on the winding engine. The other drum was provided with a winding rope, and the bucket used with this rope ran with a rider, with the pump-suspending rope as a guide at each side, and served to supply the scaffold with bricks, lime, etc. Both drums on the winding engine were worked by clutches, so that when shots were being fired at the bottom, the pulsometer and scaffold, etc., after being disconnected from the steam pipe, could be hauled up out of danger.

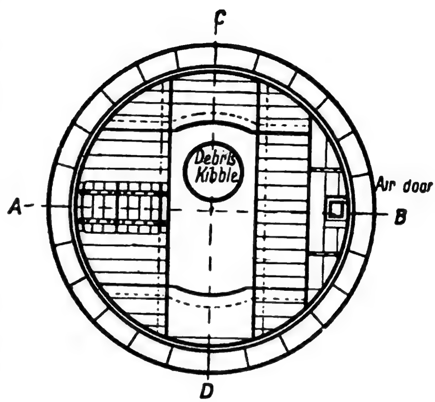


FIG. 33.—Plan of scaffold used at Newbattle Colliery.

The whole of the work and the details of sinking were carried on simultaneously with the walling without the men requiring to be drawn from the pit bottom, the only time the cradle required raising to the surface being when changing shifts.

The advantages claimed for this method are:—

It permits of quick winding with double buckets.
No time lost through sinkers being withdrawn for walling.
The brickwork can be done better by skilled bricklayers, at a less cost.
In forming lodgments or roads off the shaft-side, no delay need be incurred.
The suspension of a substantial structure in the shaft affords protection to the men working at the pit-bottom.

When the strata passed through are soft and unable to bear the weight of the curb and walling, holes are drilled all round the side of the pit, and strong iron rods $1\frac{1}{2}$ in. to 2 in. diameter and $1\frac{1}{2}$ to 3 ft. long, on which the crib is laid, are driven in. The walling then proceeds in the usual way. When these rods require to be used, the distance between the curbs should be very much shorter than in hard strata.

When the strata are giving off water, ring curbs will require to be put in, to protect the brickwork and to convey the water to a lodgment in the shaft, which is done by having a pipe connection between every two curbs.

The ordinary 'garland' or 'water ring' is usually an iron curb, cast with a groove, and is often a walling and water curb combined; sometimes it is made of wood and an annular space left in it. Above each curb the brickwork is 'shorn' back to allow the water free access to the ring (see fig. 34). It is most important that these rings should be thoroughly water-tight. Sometimes a layer of well-puddled clay is put in behind, but a better plan is to lay them on felt or oakum, and grout them with good cement.

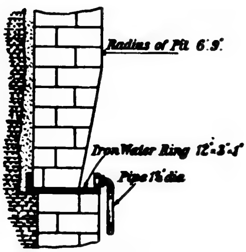


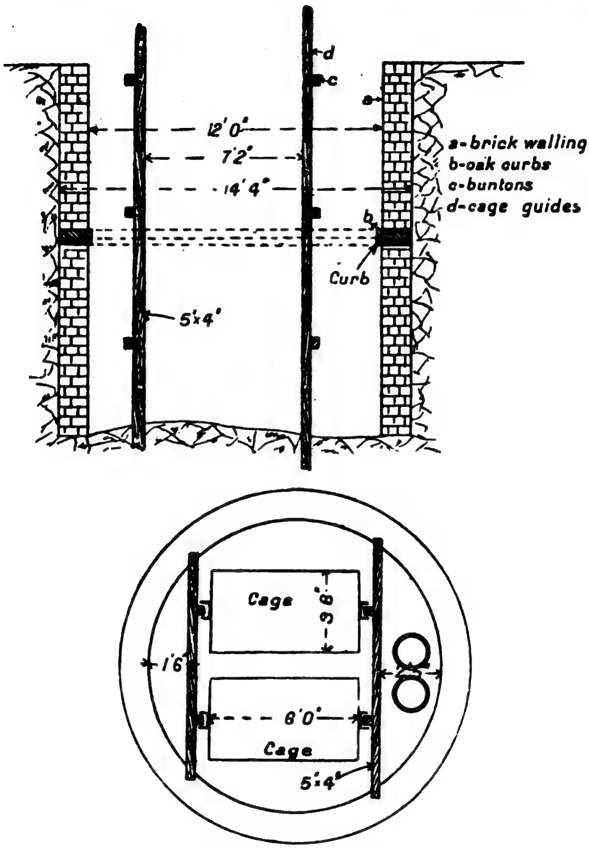
FIG. 34.—Water ring.

In walling, a good quality of brick is most necessary, the most satisfactory being good alumina fire-bricks. Except in very small shafts, ordinary shaped bricks can be used; the mortar employed should be made from good hydraulic lime, mixed with 'mine dust' from calcined iron-ore heaps, which makes a better binding material than sand.

The general arrangement of the walling, guides, etc., of a circular shaft 12 ft. diameter is shown in figs. 35 and 36, the shaft being fitted with wood guides.

Tubbing.—When large quantities of water are met with in the strata to be sunk through, cast iron tubbing is used to keep back the inflow, to save pumping and keep the shaft dry. The tubbing (figs. 37, 38, 39) is made in segments suitable to the radius of the pit, the depth and thickness of each segment varying according to the pressure to which it is to be subjected. The rings or segments are built up from a 'wedging curb,' carefully laid on a smooth bed cut round the pit. The wedging curb is a box-shaped ring of cast iron 1 in.

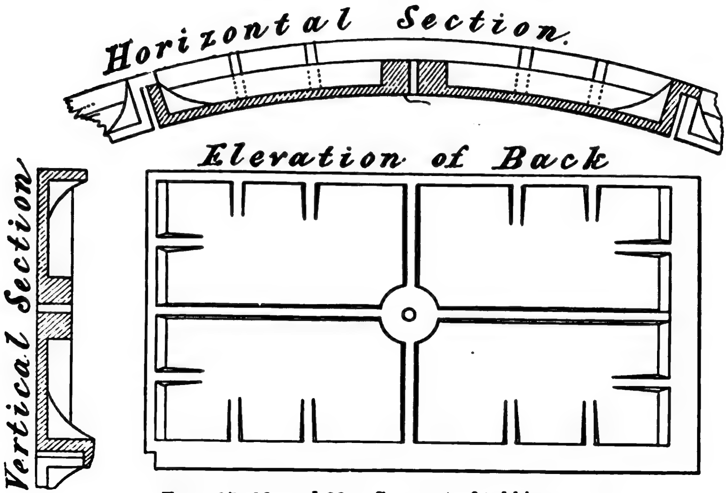
to 1½ in. thick, 6 in. to 8 in. deep, and 12 in. to 14 in. wide, and made in convenient sections. At the point where a wedging curb is to be placed, the shaft is 'shorn' back to admit its being placed in position, and a small annular space is left all round. When the curb has been laid in position and securely wedged, the tubbing is built up from it, the joints between every two rings of tubbing being filled



FIGS. 35 and 36.—Plan and sectional elevation of circular shaft.

with soft fir sheathing or thin sheet lead so as to secure a water-tight joint. The spaces left between the tubbing and the strata are usually filled with good concrete or cement. The segments of tubbing are generally cast smooth on one side, and a small hole left in the centre to relieve the pressure of water behind, while it is being built up, these holes being afterwards carefully plugged up. A corrugated

form of cast-iron tubbing has been employed on the Continent with excellent results, as it is both strong and light. The thickness of



FIGS. 37, 38, and 39.—Segment of tubbing.

tubbing required will vary according to depth and pressure to be resisted. The thickness may be found from the following rule :—

$t = \frac{RpM}{f}$, where t = thickness of tubbing in inches.

R = radius of pit in inches.
p = pressure per sq. in. in lbs. = head in ft. × ·434.
M = factor of safety (6 to 10).
f = resistance of tubbing to crushing.
(For cast iron f = 100,000 lbs. per sq. in.)

Example.—What thickness of the tubbing would be required for a shaft 15 ft. diameter, and a head of water 50 fms.?

$$t = \frac{90 \times 50 \times 6 \times {\cdot}434 \times 10}{100{,}000} = 1{\cdot}17 \text{ in.} + \tfrac{1}{8} \text{ in. (to allow for wear).}$$

N.B.—This would be the thickness required at bottom of tubbing. The formula given by Greenwell* will also supply the same information.

$$x = {\cdot}03 + \frac{PD}{50{,}000}.$$

Where x = thickness of tubbing in decimals of a foot; P = depth in feet; D = diameter of pit in feet; ·03 = a constant.

* *Mine Engineering*, by C. Greenwell.

Corrosion of Tubbing.—Certain substances held in solution by water are very injurious to iron surfaces, and to prevent corrosion, the tubbing often receives a coating of tar or of some hard varnish. In up-cast shafts where furnaces are used for ventilation, the fumes and gases given off by the furnace are also very injurious, and the only remedy is to line the tubing with good fire-brick to protect it.

Coffering.—Another method of shutting off water from the shaft is to use coffering, which is simply a brick wall with a space in the centre, this space being filled in with good cement, which makes a water-tight walling.

A third method of coffering differs somewhat from that usually employed. The wall in this case (fig. 40) is of 4½ in. brickwork, about 3 ft. at a time being built, while in front were placed sheet-iron plates *a a* 14 to 16 wire gauge.* In front of this, brickwork 20 in. thick was built up, leaving a space of 1½ in. between the inner and outer walls, to be filled in with cement, in which the iron plates were also embedded. As the pressure of the water became less the thickness of the walling was gradually reduced from 20 in. to 15 in., and finally to 10 in., the space for cement being kept in continuous lengths. The best results are obtained when the water in the pit is allowed to follow the work, so that the cement sets under water, but this is not always practicable. The brickwork in the walling should also be set in cement. At the bottom of the water-bearing strata a short length of solid walling should be put in wherever possible, and the water conveyed through it by pipes. It is generally found that when water is met with, a considerable quantity percolates through the brickwork for a few days, but as a rule it ultimately becomes quite dry. The cost of coffering of this description was, in a particular instance, £6, 6s. 8d. per vertical foot over a distance of 240 ft., which included the cost of labour and material, and the exact cost of enlarging the shaft sufficiently to admit of the extra thickness of walling. The cost for tubbing of the same length was estimated at £12 per foot for a shaft 18 ft. diameter. But £12 per foot is a very low estimate compared with some instances where this method has been adopted; as, for instance, at Shireoaks Colliery, where a total depth of 170 yds.

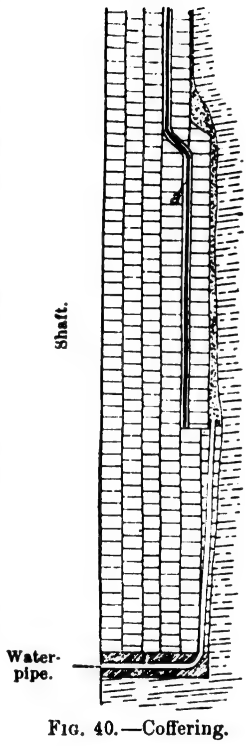


FIG. 40.—Coffering.

* *Trans. Fed. Inst. Min. Engs.*, vol. viii. pp. 18, 19.

of tubbing in a shaft 12 ft. diameter cost no less than £60, 2s. per yard.

During the sinking of the shafts at the Maypole Colliery great difficulties were experienced from extensive water feeders, one of which delivered 90,000 gallons of water per hour. The coffering arrangements, which were of a complicated and costly nature, have been described in detail in the *Journal of the British Society of Mining Students.**

The thickness of coffering required to resist a given pressure of water can be found by means of the formula given in the case of tubbing, except that a different value must be assigned to f, viz., the crushing strength of good brickwork set in cement, which may be taken as about 2500 lbs. per sq. in.

Cost of Sinking.—This will depend on the nature of the strata to be sunk through, the quantity of water to be dealt with during sinking, the size of the shaft, and the price of labour. In the particular districts labour will probably cost from 6s. to 15s. per cubic yard excavated. The cost of sinking and lining the two shafts at Harris's Navigation Colliery, each 17 ft. diameter, was as follows † :—

Average cost per yard for sinking 50 yds. in shale near pit bottom.

	Without Pumps.	With Pumps.
Labour, ...	£9 8 2 per yd.	£10 2 4 per yd.
Material (Stores, etc.), .	2 11 4 ,,	3 0 4 ,,
Total, .	£11 19 6 ,,	£13 2 6 ,,

In hard rock the cost was £44, 13s. 2d. per yard, using pumps, and the cost for walling in the same shafts, for 18 in. brickwork and two curbs per yard, was £11, 7s. 10d., or an average of £1, 3s. 10d. per cubic yard of brickwork, which seems rather high. To the above cost would require to be added the cost of guides and fixing, which would be about 25s. per yard, if iron or steel guides are used ; if wire rope guides, the cost would be 10s. to 12s. 6d. per yard.

The cost of sinking a shaft 20 ft. in diameter, 600 yds. deep, with 18 in. brickwork, in a Welsh colliery, has been given by Professor Galloway as follows :—

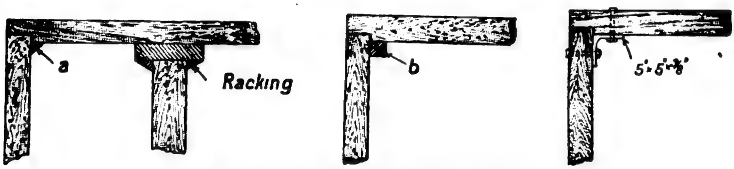
Wages and salaries, £14,400, ...	£14,400
2,500,000 bricks @ 35s. per 1000, ...	4,375
950 tons of lime @ 10s. 6d. per ton, ...	498
3500 tons of sand @ 5s. ,, ,, ...	875
600 tons of coal @ 6s. ,, ,, ...	180
Timber for mid-brattice, etc., ...	1,000
Stores, lighting, etc., ...	3,000
Contingencies, ...	3,000
	£27,328

* Vol. xxi. p. 65. † *Trans. Inst. Civil Engs.*, vol. lxiv. p. 23.

This gives an average of £46, 10s. 4d. per yard for sinking and walling alone. The rate of sinking averaged 8·3 yds. per week, and 3000 to 4000 gallons of water per hour had to be dealt with.

Rate of Sinking.—This varies very much according to the strata and difficulties met with, ranging, for a shaft 15 ft. to 16 ft. diameter, from 2½ to 3 yds. in very hard strata to 8 or 10 yds. in ordinary strata per week.

Sinking Rectangular Shafts.—The procedure is much the same in sinking rectangular shafts as in the sinking of circular ones. The shaft having been properly pegged off, the surface soil is excavated, as already described, and the sides supported with temporary wood, until a convenient depth or the rock head is reached, when the first set of 'barring' is usually put in, great care being taken to square the bed for it, and to set it level. The first set having been properly adjusted, others are built up above it, to 3 ft. or thereabouts above the surface, to afford sufficient height for emptying the material excavated, and also to prevent water flowing into the shaft,



FIGS. 41, 42, and 43.—Fixing timber.

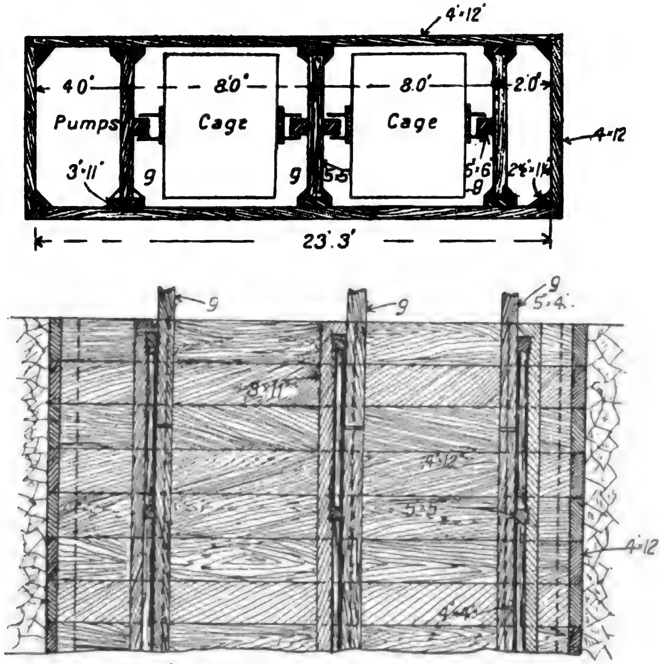
the back of the barring being well puddled with good blue clay for this purpose.

The sets of barring are fitted into the shaft, either cut square, with an ordinary 'butt' joint, and corner rackings (fig. 41) put in to bind them together; or they may be notched into each other (fig. 42), which makes a neater and stronger job. Corner rackings are also used square as at fig. 42, and angle-bars are also occasionally employed for this purpose (see fig. 43). They are neat and strong; and they have the further merit of lasting very much longer than wood. The barring when put in position should be well and tightly wedged at the corners, and also opposite each bunton; the spaces behind the barring should be well packed with some light material, branches of fir trees for preference, to ensure efficient drainage. Figs. 44 and 45 show the plan and elevation of a rectangular shaft, and illustrate how the lining, buntons, etc., are fixed. The sizes of wood used for barring (lining) vary according to the nature of the strata passed through. In ordinary strata not giving off much water, barring of white or red pine, 9 in. × 4 in., is used at the surface, and 9 in. × 3 in. in the rest of the shaft. Where the pressure is great and the shaft large, or where loose material has to be passed through, the barring may be 9 in. × 5 in. or 12 in. × 6 in. The corner rackings

are usually $2\frac{1}{2}$ in. × $1\frac{1}{2}$ in. or 2 in. square; if angle iron is used it may be 5 in. × 5 in. × $\frac{3}{4}$ in., or 5 in. × 4 in. × $\frac{1}{2}$ in.

Wood lining in down-cast shafts lasts, on an average, about fifteen years, but in up-cast shafts the average is much shorter.

When the barring is fitted in it is further strengthened by wall-plates and buntons, the former being put opposite each bunton, and the latter themselves being put in at right angles to the barring, the perpendicular distance between them varying from 3 ft. to 6 ft., according to the strata passed through, but averaging 4 ft. The



FIGS. 44 and 45.—Plan and elevation of rectangular shaft.

buntons may be of either white or red pine; the sizes used are 5 in. × 5 in., 6 in. × 6 in., or 8 in. × 6 in., or for small shafts 8 in. × 3 in.

It is now the general practice, especially in large shafts, to put 'filling-in pieces' or 'punch props' between the buntons at each end, and also at the centre of each of the latter, these 'punch props' giving greater strength and stability.

Rectangular shafts are now generally lined from top to bottom. This gives additional security to the shaft, and facilitates the fixing of the buntons and guides.

After the surface soil has been sunk through it is usual to erect a

windlass or steam crane, but a windlass is only suitable for small shafts, and can only be economically used for depths of 15 or 20 yds., beyond which it is better either to employ a small temporary sinking engine, or to erect the permanent winding engines at once. For large shafts steam cranes are much used for the earlier sinking, as they give more power than a windlass, besides being speedier and safer to work with, while the bucket can be swung clear of the shaft, and landed at any desired point for tipping.

If a temporary engine is used for sinking, it should be placed in such a position as not to interfere with the erection of the permanent winding engine, otherwise much delay may be caused. The temporary engine is often erected as close to the shaft as possible, so that the permanent engines may be laid down behind it, and the erection of screens, etc., may be proceeded with while sinking is going on. This saves time and enables coal to be dealt with immediately the shafts are sunk. The sinking engine is sometimes placed in such a position that it can be afterwards used for haulage purposes underground.

When sinking, the shaft is usually covered over, only sufficient space being left for the bucket to pass through. When the kibble is tipped at the top of the shaft without the aid of scaffolding, a strong beam is laid across the pit, to which 'sliding' deals are fixed, to prevent the bucket from catching the mouth of the shaft, and also to make it 'strike' easier if no bogie or chain is used. Very often a bogie is used for receiving the kibble when it arrives at the surface, made so that it entirely covers the shaft (fig. 46), and prevents anything from falling on the men at work at the bottom. Sometimes a chain or rope fixed to a beam on the pithead frame is used to swing the kibble clear of the shaft. Another method of closing the top of the shaft is by means of folding-doors with rails on their upper sides. The accompanying illustrations (figs. 47 and 48) show the arrangement used by Professor Wm. Galloway while sinking the Llanbradach shafts.* The two folding wooden doors are held together by hinges a' a', which are keyed on to shafts b b'. Balance weights, c c c' c', are attached (two to each door), and these are connected by rods d d', through cranks on the two opposite shafts so that the doors open and shut simultaneously when the hand lever g is drawn backward or pushed forward respectively; e e' shows the position of doors when open, and the balance weights f f will then be in the position shown in fig. 47. If the doors are shut when the winding rope is in the shaft, the two guide ropes and the winding rope pass through three holes on the centre line of the door. A beam is put across the shaft directly below the balance weights, which are boxed in to prevent the possibility of any accident. The rods, levers, cranks, and balance weights are also boxed in above, and only the lever g projects through longitudinal slots in the cover. In using

* *Lectures on Shaft Sinking*, p. 7.

this apparatus, when the bucket is at the surface, the doors are closed, and a tipping waggon into which the contents of the bucket are emptied is run on, without taking the bucket off the winding rope. The waggon is then withdrawn, the doors opened, and the bucket is ready to descend. Whatever method is adopted, great care must be taken to let no loose material, such as stones, bolts, etc., fall down the shafts.

Preparing the Wood.—All wood, such as barring, buntons, racking, etc., should be prepared at the surface, ready to be sent down the pit as required, as this saves much labour, wood being difficult to cut and dress in a confined shaft. A hand winch, with a thin wire rope

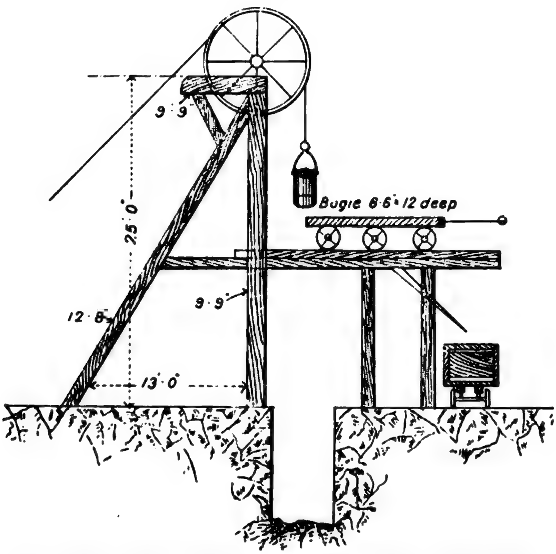
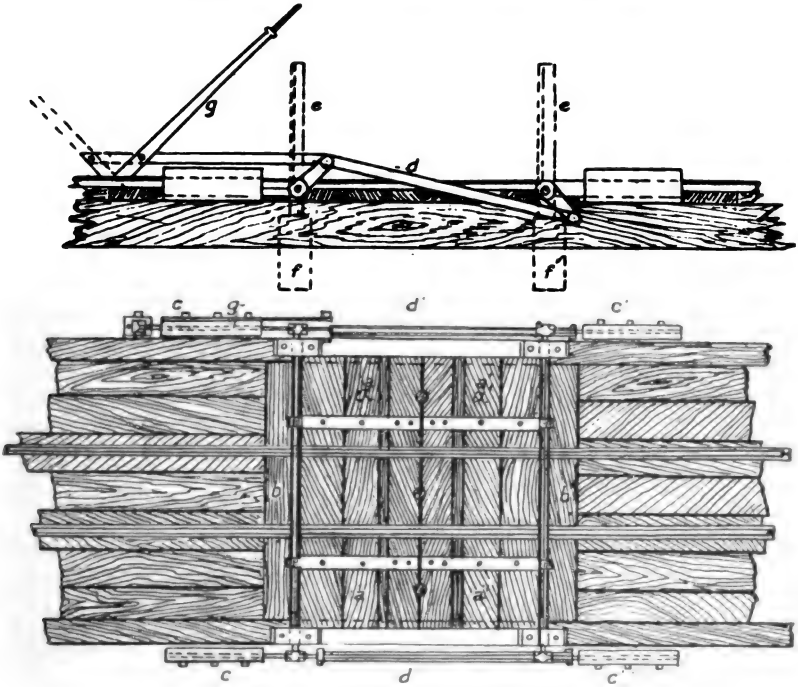


FIG. 46.—Arrangement for receiving kibble with bogie.

and a large muzzle attached, should be kept ready for lowering the wood to the sinkers as required.

Disposition of Labour and Tools Required.—The sinking is generally carried on continuously during the twenty-four hours, with the exception of Sundays, and the number of men employed on each shift varies according to the size of the shaft, etc. For a rectangular shaft 23 ft. × 7 ft. the number of men employed would be twenty-one, *i.e.* seven men on each shift; for smaller shafts three or four men on each shift would be sufficient. In a circular shaft 18 ft. to 20 ft. in diameter, sixteen to twenty men should suffice for each shift; an average labour allowance being one man for every 18 to 20 square feet of sinking.

The tools used in sinking are spades, shovels, picks, jumpers, 2 ft., 3 ft., and 4 ft. long, and 1 in. to $2\frac{1}{2}$ in. across the mouth; single and double-headed hammers, stemmers, cleaners, saws, axes, screw-keys, and porting-bolts. Two kibbles will also be required, each to



Figs. 47 and 48.—Galloway system.

hold 10 to 20 cwts. of material, and also a water-barrel to hold 15 to 30 cwts. of water.

Special Methods of Sinking.—When thick beds of running sand, gravel and water, or peat moss or mud and boulders, are met with either at the surface or further down, the following special methods of sinking may be employed:—

Sinking by pile driving.
Sinking by brick drum, iron or steel cylinders, rectangular iron cylinders.
Sinking by combination of brick drum and iron or steel cylinder.
Sinking by compressed air; *e.g.*, Triger system.
Sinking by boring or drilling out the shaft; *e.g.*, Kind-Chaudron system.
Sinking by freezing the strata; *e.g.*, Poetsch or Gobert systems.

Pile-driving, or sinking by piles, is one of the commonest and easiest methods of sinking through a moderately deep bed of sand met with at the surface.

The piles used for this purpose are usually of red or white pine 12 to 15 ft. long, 9 in. broad, and 3 in. thick, sharpened and shod with iron at the bottom, to facilitate driving, while at the top a hoop of iron is shrunk on to prevent splitting while the pile is being driven down (see fig. 49). Before starting to sink, a strong framework of timber, of the size required, is fitted together, and laid down on the site where the shaft is to be sunk. The first set of piles are then driven in all round it 'skin for skin,' the commonest method of driving them being by hand, the man using a large mallet. If they cannot be driven easily by the mallet or hammer, a 'monkey' may be used, or the necessary pressure applied by means of a hydraulic ram. As the piles of each succeeding set are driven in they are firmly supported by side and end bars, and buntons placed at convenient distance apart, as shown in fig. 49. When a pit has to be sunk by this method, it must be commenced very much larger than the finished dimensions required, especially if the depth of sand to be sunk through is considerable, as every set of piles put in reduces the size of the shaft by at least 18 in. Sometimes the piles are driven inclined outwards to keep the size of shaft from being reduced too much, but by this method it is more difficult to keep the barring perpendicular. If any space is left between the piles and the walling or barring, it should be filled up with good cement or concrete, and the piles withdrawn if possible. Sinking by piles is an expensive method, and sometimes not a very successful one, if the sand is very quick, or when the strata are watery and mixed with boulders. The limit of depth that can be sunk through by pile-driving is about 60 or 70 ft., but it is more efficient when the depth does not exceed 30 to 40 ft.

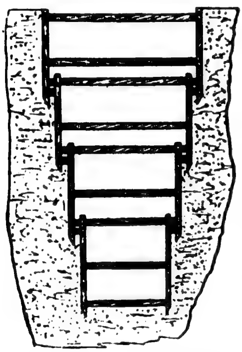
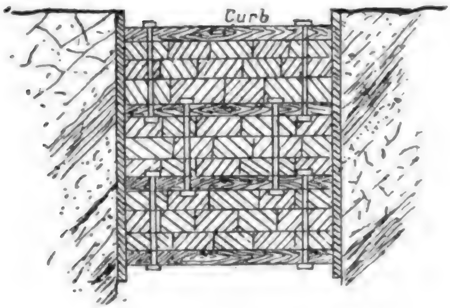
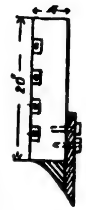


FIG. 49.—Pile-driving.

Brick Drums.—Sinking through running sand is often done by what is known as the 'Drum' method. When this system is adopted, a curb of wood, 14 in. to 18 in. broad and 6 in. thick (figs. 50, 51), is laid down on the site to be sunk through; in a rectangular shaft a square frame is used, firmly morticed and bolted together. The curb or frame is carefully adjusted with a straight-edge and spirit-level, to get it perfectly horizontal. Upon this curb a tier of dry brickwork is placed, until a height of 3 or 4 ft. has been reached, when another curb is placed in position and secured to the first by strong tie-bolts of wrought iron; more brickwork is then placed in position until the drum begins to sink by its own weight. Workmen stand in the centre and excavate the sand as it sinks, taking care that the bottom of the drum is 2 or 3 ft. in advance of the excavation, and at the same time keeping it in a horizontal position, for one of

the great difficulties in sinking by this method is keeping the drum truly vertical, so as to prevent it from canting. To reduce the friction of the drum during descent, and also to keep the brickwork intact, a close lining of planking is fixed all round its outer circumference, the joints being made water-tight. As the sinking proceeds more brickwork and curbs are placed in position, and





FIGS. 50 and 51.—Brick drum. Cutting shoe.

secured by tie-bolts as before, until the solid ground is finally reached, where a perfectly level bed must be made for the reception of the first permanent walling curb. If the ground is not loose enough for the drum to sink easily, a cutting edge is fixed to it, bevelled on the inside and fitted with an iron shoe (fig. 51). For the purpose of keeping the drum plumb and sinking evenly, it is sometimes lowered

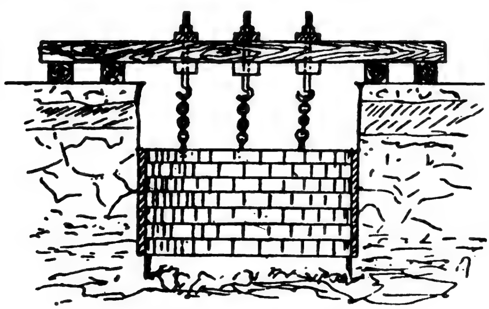


FIG. 52.—Hanging brick drum.

by strong screws and nuts attached to beams at the surface, or it may be lowered by ropes and winches. This method affords better control over the operations (fig. 52).

Cast or Wrought-iron Cylinder.—Brick drums are liable to stick, and iron cylinders or drums are therefore preferable. These iron drums are made in segments, cast to the curvature of the pit

and strengthened by horizontal and vertical ribs, like ordinary tubbing, with the exception that the ribs are cast on *the inside* so that the outside of the metal is left smooth, and offers as little resistance as possible when sinking through the sand. The joints are carefully rendered water-tight by putting sheet lead between the flanges and firmly bolting them together, a cutting edge being attached to the bottom in much the same way as to the brick drum.

In Scotland, a method of sinking rectangular shafts through running sand with iron tanks is adopted which is somewhat similar to the above system. The tanks are usually made of ordinary boiler-plate fastened together by lap joints and rivets.

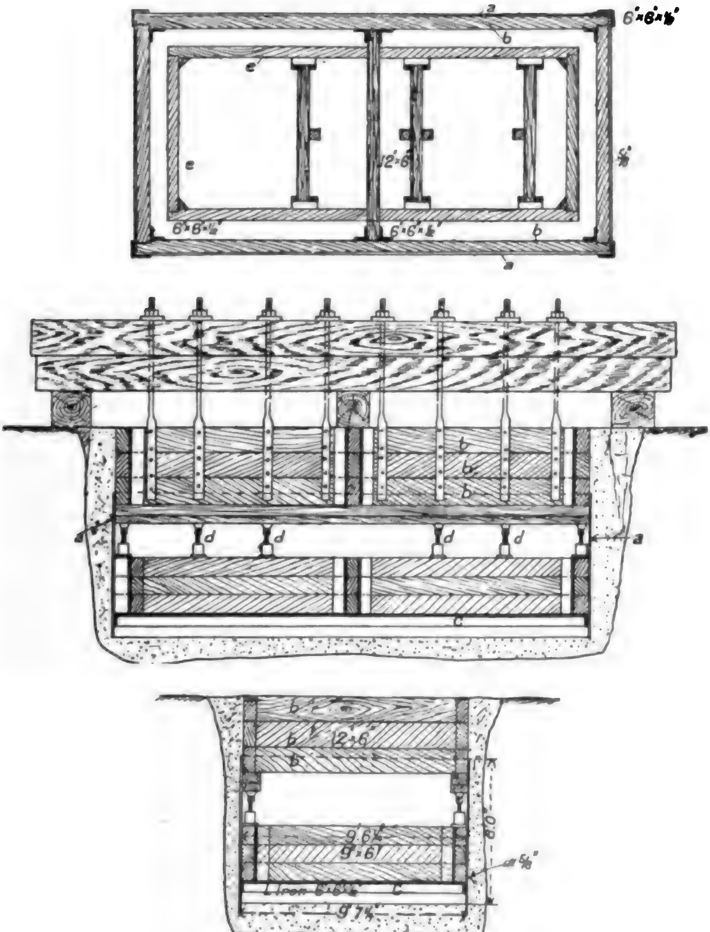
Figs. 53, 54, 55 show the arrangement of the tank. The tank is made of segments of wrought-iron boiler plate $\frac{1}{2}$ in. or $\frac{5}{8}$ in. thick, and 6 ft. deep. The segments are firmly joined at the corners with overlapping pieces, with which they are riveted 'flush.'

When about to sink, the drum is set in the position of the proposed shaft, and the sand or mud removed from the inside, until a sufficient depth has been reached to put in two or three sets of barring, *b b*, of 12 in. × 6 in. pitch pine. The tank is then forced down into the sand by means of screw-jacks. Near to its foot is an angle iron to which three sets of oak barring 9 in. × 6 in. are fixed, for the purpose of giving a seat to a number of screw-jacks which are used to lower the tank as the sinking proceeds. When the first three sets of barring at the top have been fixed to strong beams placed across the shaft (fig. 54) by means of hangers and nuts, the tank is pressed down a further distance of a foot or so, and the sand dug out until room has been made to add another set of barring below that already fixed, to which the last set is hung by means of wrought-iron straps and corner angle irons. This operation is continued until the tank has been sunk its full depth of 6 ft., when the same procedure is gone through as before, pressing down the cylinders by means of the screw-jacks and adding barring at the bottom as required until the bed of sand has been sunk through.

By this method beds of running sand of almost any depth can be sunk through with safety and rapidity. The tank has, of course, to be made a great deal larger than the finished size of shaft, as will be seen from fig. 53. When the rock head has been reached, the regular barring is built up to form the shaft inside the first temporary barring, and a space, which may be filled in with cement or concrete, left all round. In sinking either with this or any other sort of drum, the greatest difficulty is to keep it vertical, and this can only be done by maintaining a careful watch on the screws when lowering the drum and removing the sand. It is better to make the drum a little wider at the bottom, 1 in. or 2 in., than at the top end, as this will assist it to descend more easily than if it were the same width throughout.

Combination of Brick Drum and Iron or Steel Cylinder.—In found of great advantage in reducing 'skin' friction and giving

sinking through running strata with the brick drum alone it has been found, as already stated, that there is always a considerable tendency for it to go off the perpendicular, to stick fast altogether, and to give



FIGS. 53, 54, and 55.—Iron tanks.

a = iron cylinder ; *b* = outside barring ; *c* = angle iron ; *d* = screw jacks ; *e* = inner barring for shaft.

an excessive amount of 'skin' friction. To overcome these difficulties, especially in very soft surfaces, a steel cylinder is now frequently used in conjunction with the brickwork. This arrangement has been

additional strength to the drum, and making it sink more easily. Two shafts have recently been sunk on this principle under very difficult conditions at Olive Bank Colliery, near Edinburgh, and as they form a typical example of this system of sinking, they may here be described more fully.

There are two shafts, 70 ft. apart, each 14 ft. in diameter inside the finished brickwork. The pits have been sunk to a depth of 152 fms., and having the following section at the surface: sand, 5 ft.; gravel, 5 ft.; boulder clay, 22 ft.; silt or mud, 75 ft.; red sandstone (with much water), 40 ft.; total, 147 ft.

A square pit was first sunk through the boulder clay until the top of the running mud was reached. This pit measured 18 ft. 6 in. inside the wood, and was lined with 9 in. broad × 4 in. pitch pine. When the pit was secured in this way down to the top of the mud—*i.e.* to the bottom of the boulder clay, a distance of 32 ft.—the bottom segment of the steel cylinder with the cutting edge was built in and fixed in position. On the top of this segment were built up as many rings of the cylinder as brought it above the surface level, the work of lowering the rings being carried out by means of a steam crane. Before the actual sinking through the mud was entered upon, the cylinder was therefore between 30 and 40 ft. in length. The cylinder was built entirely of steel plates, the bottom segment being 5 ft. deep and $\frac{3}{4}$-in. thick, while the other rings were 4 ft. deep and $\frac{1}{2}$-in. thick. The outside diameter of the cylinder was 18 ft. $2\frac{1}{2}$ in., and the circle made up of twelve segments. To facilitate the sinking of the cylinder the diameter of the bottom ring was 18 ft. 4 in., $1\frac{1}{2}$ in. wider than the rest of the cylinder. The segments were joined together by means of **T**-pieces, 6 in. × 3 in. × $\frac{1}{2}$ in., and at the four joints opposite the centre lines and covering pieces, 6 in. × $\frac{1}{2}$ in. were also used. At the top of the bottom segment a projecting piece was built all round the inside of the cylinder as a foundation for the brickwork which was to form the shaft lining. This projecting piece was supported by brackets, also made of **T**-pieces 6 in. × 3 in. × $\frac{1}{2}$ in., and fixed at intervals round the circumference of the cylinder. To give further support to this projecting shelf which carries the walling, another projection, consisting of angle-irons, was fixed on the bottom of the lower segment, and from this projecting piece a tapered section of brickwork was carried up to the shelf above, on which rests the regular walling. At the outer edge of this upper shelf or support, an angle-iron, 3 in. × 3 in. × $\frac{1}{2}$ in., was fixed at a distance of 2 ft. from the inside of the cylinder, and this formed the circle for the shaft lining. The lining consisted of an outer ring of 18 in. of brickwork, and the remaining space of 6 in. was filled in with a mixture of lime and pure cement. When the cylinder was placed in position and the sinking through the mud about to start, the procedure was to dig out the silt or mud to a depth of 2 or 3 ft. at a time. If the cylinder would not sink by its

own weight, more brickwork was added from the top, the brickwork lining being gradually added to until the surface was reached. When the cylinder would no longer sink by its own weight *plus* the weight of the walling, which was found to occur at a depth of 75 to 80 ft. where the 'skin' friction became excessive, additional weight was added by using pig iron placed on a scaffold resting on 9 in. × 6 in. pitch-pine buntons built into the walling. Before the mud was finally sunk through, the total weight of pig iron resting on the cylinder amounted to 400 tons, and the combined weight of cylinder and pig iron exceeded 800 tons. The digging of mud and the loading of the cylinder were continued till the rock-head was reached. The progress made in the sinking varied greatly from 3½ ft. down to ½ in. per twenty-four hours. The total time taken to sink the cylinder through the mud and boulder clay down to the rock was about five months, this time including the fixing of the rings, building the walling, and all the other work connected with the sinking.

Triger's Method.—This system was first applied by a French engineer, M. Triger, about the year 1841, to sink a shaft on an island in the Loire.*

The system essentially consists in forcing iron tubbing down through the ground by pressure applied from above, and in furnishing the tubbing with an air chamber, which forms a double diaphragm interposed between the outside atmosphere and the interior of the pit, and maintains therein a pressure equal to that due to the head of water at the lower end of the tubbing.

Figs. 56 and 57 show the arrangement of tubbing, etc. A is the malleable-iron chamber, with two man-holes D B. When placed in position at the mouth of the pit and firmly secured, it admits of further excavation for the introduction of the cutting ring of the tubbing C. Compressed air is forced in through the pipe P into the lower compartment, and the air pressure being greater than that of the water contained in the sand or strata, it holds in check any water tending to flow in, or forces it up through a flexible tube.

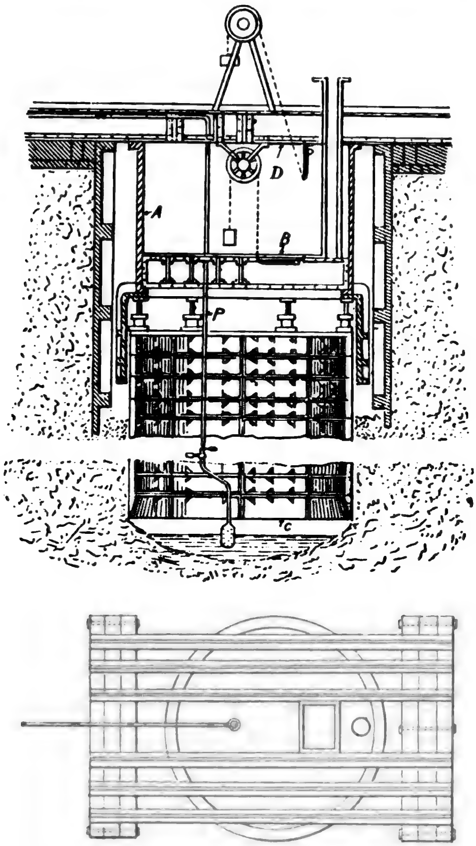
The workmen enter the air chamber by a door D at the top, and as soon as they are in, the opening is immediately shut. When the pressure of air in this chamber becomes equal to the pressure in the lower compartment B, the second or lower door is opened, and the workmen proceed into the interior of the pit, and there carry on the operations of sinking, of forcing down the tubbing by means of screw-jacks, and of adding the segments as required.

All the doors and joints should be as nearly air-tight as possible, and the doors leading to the compartments are never both opened at the same time. The maximum depth that can be sunk by this method is about 100 ft., the air pressure at that depth being equal to three atmospheres, or 45 lbs. per sq. in. Even then, it is difficult to get workmen to stand it without injury to health. At

* *Trans. Min. Inst. Scot.*, vol. vi. p. 27.

Puits Marie, near Aix-la-Chapelle, however, a shaft was sunk to a depth of 111 ft. in water strata by this system.*

In this instance, it was anticipated that the capillarity existing between the particles of running sand would have the effect of lightening the pressure due to this head of water, and the facts



FIGS. 56 and 57.—Triger's method of sinking.

justified the assumption. A good many shafts have been sunk by this system, as also foundations for piers of bridges, etc.

Like the Kind-Chaudron method of sinking, it is very expensive, and may cost anything between £80 and £800 per foot sunk.

Kind-Chaudron Method.—In some districts in England, the coal-measures are overlaid by strata containing very large quantities of

* *Trans. Min. Inst. Scot.*, vol. vi. p. 23.

water, which would entail great expense in pumping during the sinking. In some parts of the French coal-fields, the coal-bearing measures are overlaid with chalk, which likewise contains large volumes of water, the coal strata below being comparatively dry. To sink shafts is, in either of these cases, very expensive, and heavy pumping machinery would be required while sinking through the water-bearing strata in the ordinary method, while after the coal-beds were reached and tubbing put into the shaft, no pumping apparatus would perhaps be required, and much valuable machinery would be left on hand that would be of little use, and could only be sold at considerable loss. It was to successfully meet and overcome such difficulties that the Kind-Chaudron system of sinking was introduced and adopted. Since its introduction over eighty shafts have been sunk on this system, in various parts of Europe, including six in England. The sinkings at Dover are being carried out on this system.

Briefly, this method consists of boring out the shaft, and then lowering into it a water-tight lining of cast-iron tubbing.

This system of sinking may be divided into the following stages * :—

Alternately boring a small pit in advance and then enlarging it by a larger tool to the full size of the shaft.
Preparing a seat for the 'moss-box.'
Lowering the water-tight lining or tubbing with the moss-box at the bottom.
Putting in the outside lining of concrete.
Pumping out the water.

In the preliminary operations, a small pit 4 ft. to 8½ ft. in diameter is bored out by a tool known as the small trepan, and weighing about 8 to 12 tons. It is supplied with 14 cutting teeth or chisels of chilled steel, securely fastened into the jaw of the trepan (figs. 58, 59, and 60 show details of small trepan). The trepan is suspended by pitch-pine rods 7 in. to 8 in. square, and in long lengths of 50 or 60 ft. The rods or spears are actuated by a steam-engine with a vertical cylinder of 30 to 40 in. diameter and a stroke of about 4 ft. A large strong beam with the fulcrum nearer the pit than to the engine, is attached at one end to the piston in the cylinder, and at the other to the rods to which the trepan is fixed. The steam-engine actuates the trepan through this beam in much the same way that bore rods are worked by a brake-staff, raising it from 1 to 2 feet at every stroke and then allowing it to fall sharply by its own weight, the rods being turned in the usual way after each stroke by means of a cross-piece or 'tiller.' The débris in the small pit is removed by a sludger, which can be either attached to the rods or let down by a wire rope wound on a drum worked by a small horizontal engine for the purpose.

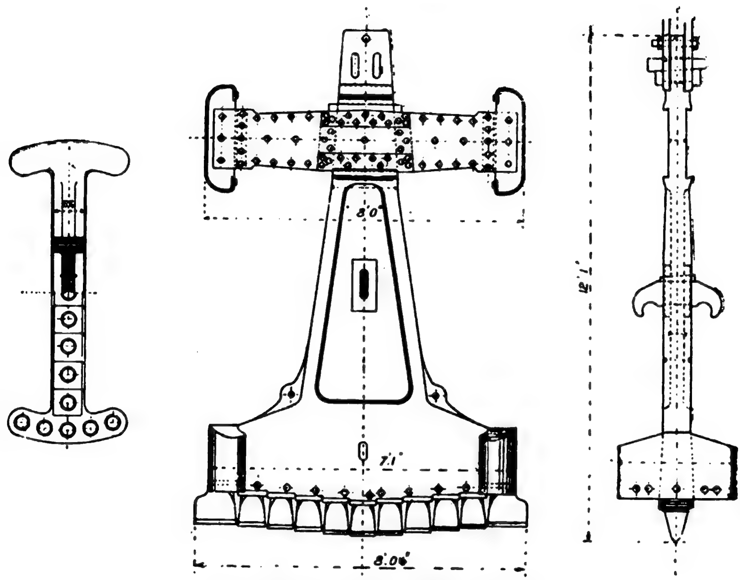
When the pit has been bored out for a depth of 20 or 30 yds., the small trepan is withdrawn and the larger tool is set to work.

This large trepan weighs about 16 tons, and is fitted with a strong

* *Ore and Stone Mining*, C. Le Neve Foster, sixth edition, p. 288.

iron bow in the centre which is a little smaller than the diameter of the small pit, into which it fits, acting like a guide. The teeth, about nine in number on each side, are fixed at each end of this bow, those near the centre being longer than those at the extremities (fig. 61), the object being to make the upper edges of the larger pit slope toward the small pit, which has already been bored, and so facilitate the free passage of débris into the receiving bucket or pan. This bucket or pan is inserted at the bottom of the small pit, and the refuse falls into it, as it is cut by the large trepan.

An iron bow is provided on this bucket which can be caught by a



FIGS. 58, 59, and 60.—Small trepan.

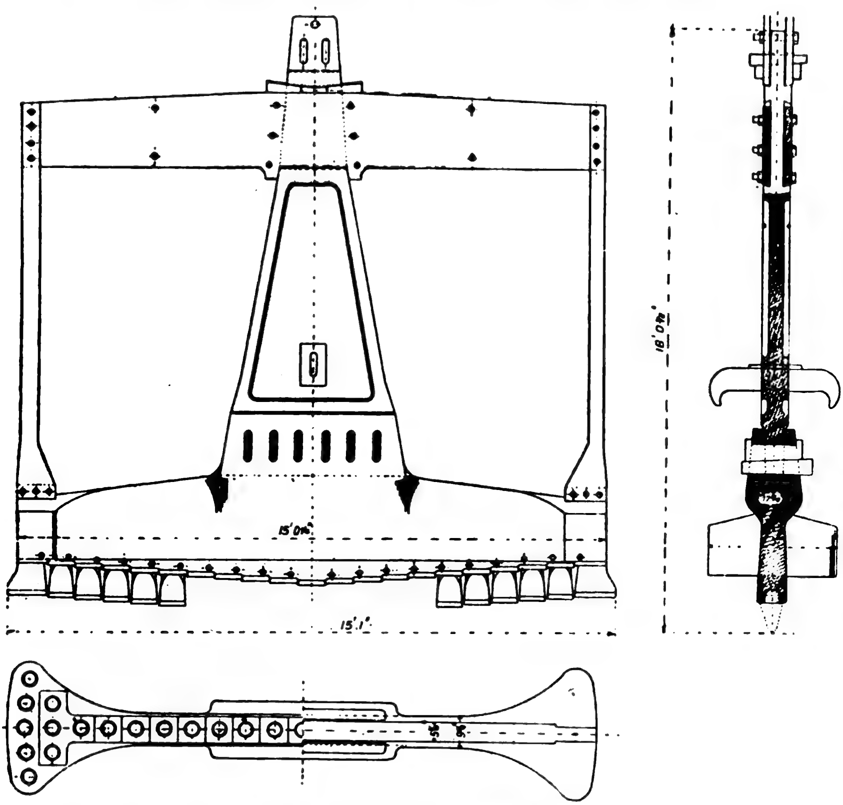
grapnel attached to the rope, and thus raised to the surface. Figs. 61, 62, and 63 show detailed drawings of the large trepan. When the sinking has reached the firm rock a smooth bed is carefully prepared for the tubbing and the moss-box to rest on, for upon the tightness of the tubbing depends the whole success of the process.

This bed is made level by a special tool somewhat resembling a large pair of 'lazy-tongs.'

When the shaft has been sunk to the required depth, the most difficult part of the work, viz., the lowering and fixing of the metal tubbing, is then proceeded with. This consists of cast-iron rings *a a* (fig. 64), the full diameter of the finished pit, each ring being about 5 ft. deep and varying in thickness according to depth.

They are cast with internal flanges, and the rings are joined to one another by bolts *b b*, the joints being made water-tight by the insertion between them of thin lead sheeting. At the bottom of the tubbing are two rings with flanges turned outwards and so arranged that they can slide over each other (fig. 65).

The space between these two flanges is filled with moss, which,

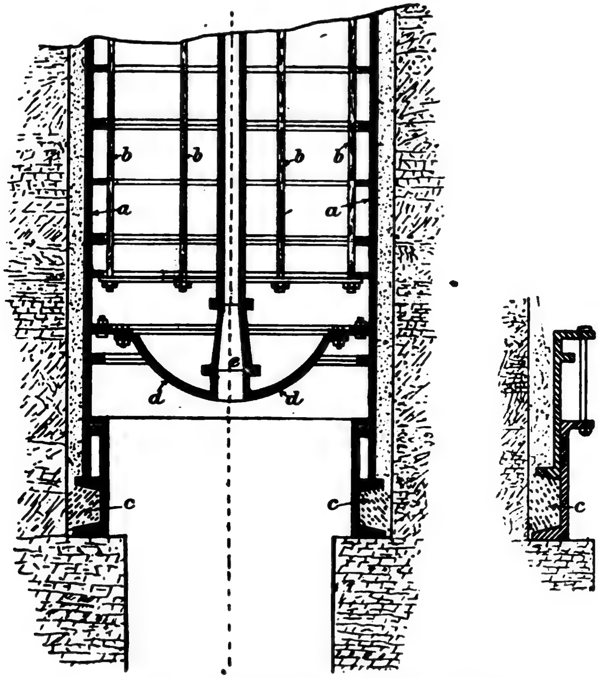


FIGS. 61, 62, and 63.—Large trepan.

when compressed by the weight of the tubbing as it is lowered into position, makes a water-tight joint.

Immediately above the moss-box *c* a false curved bottom *d* is bolted on, with a tube *e* in the centre, which allows of rods being worked through it and the pressure of water to be lessened as the tubbing descends. The whole column of iron tubbing, with moss-box attached, is lowered by strong iron screws and rods

attached to heavy beams, placed over the shaft at the surface or to beams fixed in the shaft above the water-bearing strata. When the column is lowered, the space left between the outside of the tubbing and the side of the shaft is carefully filled in with good cement or concrete, lowered in boxes so constructed that their contents can be discharged at any definite point. After ample time has been allowed for the cement to 'set' and harden, the water is drawn out of the shaft by means of a water-barrel or pump, and the rest of the sinking is then proceeded with in the ordinary way.



FIGS. 64 and 65.—Kind-Chaudron method.

By means of the false curved bottom or diaphragm mentioned above, the pressure can be so adjusted that the tubbing can be practically floated into position, thus reducing the strain on the lowering rods to a minimum. Of course, at great depths where, owing to the pressure, the metal tubbing requires to be very heavy, there must be, of necessity, a great strain on the lowering rods. It is evident, therefore, that the method of installing the tubbing will depend largely on the depth, and will have to be adjusted to meet special cases.

Within the last few years several modifications of this system of sinking have been introduced and carried out with success. The moss-box is no longer considered necessary in the fixing of the tubbing, reliance for a water-tight joint being placed in making a carefully prepared bed and good cementing.

At a shaft sunk by the Liévin Company in the north of France, the shaft was bored out in two operations; a first pit 6 ft. 6 in. wide being sunk to a depth of 10 or 12 yds. beyond the watery strata, which was subsequently enlarged by a second boring to the full size.*

On reaching the required depth the teeth of the trepan are set so as to cut a horizontal and level bed for the tubbing to rest on, and the use of moss-box, equilibrium tube, and false bottom can be entirely discarded. Work can, by this method, be carried on with great rapidity; in one instance the small shaft was bored out to a depth of 366 ft. in seventy-five days, the larger one being bored out to 327 ft. in four months twenty-one days, and fixing the tubbing occupied two months longer. The ordinary rate of boring by this system is 9 to 12 in. per day, according to the depth and diameter of the shaft.

Lippman's Method.—This method of sinking is practically the same as the Kind-Chaudron, but instead of the shaft being bored out in two or three operations, it is completed in one, *i.e.* the shaft is bored out from the commencement with a large trepan specially made for the purpose.

Advantages of these Systems.—For this and the Kind-Chaudron method of sinking the advantages are † :—

The use of pumps is avoided, unless when the shaft requires to be cleared after the tubbing has been lowered.

The risk of accidents to workmen, which are common in the ordinary mode of sinking, are reduced.

The inconvenience of draining the surrounding springs, which in a populous district depending on these for a water supply would be great, is avoided.

Shafts may be sunk to coal seams through ground which it would be impossible to deal with by the ordinary methods of sinking.

Against these advantages there must, however, be set the fact that these methods of sinking are very costly.

The cost of such sinkings may vary from £50 to £150 per yard depth, according to the strata and the difficulties encountered. This price does not, of course, include the cost of the tubbing and other accessories.

Pattberg System.—This system, like the Kind-Chaudron, is applied for drilling out shafts in water-bearing strata containing large quantities of water. It somewhat resembles the Kind-Chaudron method, but has several distinctive features which are quite new.

* *Ore and Stone Mining*, sixth edition, p. 293.

† *Trans. Min. Inst. Scot.*, vol. vi. p. 28.

Two shafts have recently been successfully sunk by this method at the Rheinpreussen Colliery, near Homberg, Germany.

The principal appliances used in this method of sinking are the percussive boring tool mounted on a strong wooden or wrought-iron frame, and supported by a tubular boring rod (see figs. 66, 67), and two mammoth pumps, the whole being slung from a scaffolding over the shaft, and an oscillating drum, driven by a steam engine, for giving reciprocating motion to the cutter. The borer B (fig. 67) hangs on a wrought-iron tubular boring rod, having an inside diameter of 150 mm. (6 in.) and 15 mm. ($\frac{3}{8}$ in.) thickness of metal. The chisel-carrying part *r* is also of wrought iron. It slopes upwards from the centre to both sides, so as to cut a surface inclining towards the centre of the shafts, and has on either side a tube-like piece *a*, from which the small channels *b b* branch off at

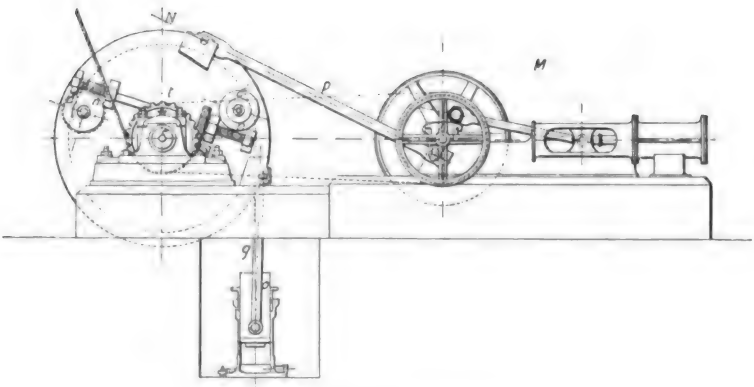


FIG. 66.—Pattberg System.

right angles and lead into the corresponding channels in the steel chisel teeth *z z*. The tubular boring rod, from which the cutting tool is suspended, is in communication with the hollow pieces (*a a*), and supplies the water which flows out at the edge of the chisel teeth. The vertical and horizontal guiding arms *u* and *v*, as well as the other supporting pieces, are made of wood. The apparatus which was first employed had a cutting edge of 6·4 metres (20·99 ft.) broad and 8·3 metres (26·89 ft.) high in the centre, the total weight of the boring piece being 19,800 lbs.

Instead of the screws hitherto used for holding the individual parts together, wedges were used for this new borer, as it was thought that owing to the large number of times it had to be raised and lowered, the screws would very soon become loose. Further, instead of constructing this borer with teeth, a straight cutting edge was put on.

At either side of the tubular boring rods is one mammoth pump. These mammoth pumps consist of two pipes (R R, fig. 67) of 3 mm. (·117 in.) and 140 mm. (6·5 in.) inside diameter. The pipes reach down almost to the point of the borer, and enclose a second pipe of the same thickness and 100 mm. (3·9 in.) diameter. In the annular space between the two pipes compressed air is brought from the surface, which is allowed to escape a little above the lower end into the inner tube. This causes a pressure on the surface of the detrital sludge, causing it to be sucked off the centre of the shaft and brought to the surface.

The tubular boring rods are led into the shaft scaffolding through a hollow guide, and have on the top a revolving piece to which a rope is fastened. This rope is wound round an oscillating drum (*t*, fig. 66), which is operated by a steam engine through a crank shaft *p* connected to a large drum or disc N. In order to release the boring arrangement the weight hanging on the rope is partly adjusted or counterbalanced by steam pressure, by means of a plunger, connected by a rod *g* to the drum N.

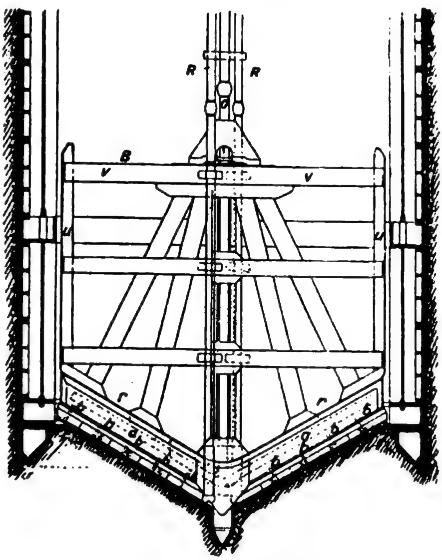


FIG. 67.—Pattberg Cutter.

About 40 horse-power are required for starting the boring arrangement. While boring is proceeding, the alternate slackening and tightening of the rope, to give percussion action to the cutting head, is effected in the following manner:—The drum (fig. 66) revolves on the axle *e* of the boring apparatus, drum *t* and another disc behind I are fixed to the axles by wedges. The disc behind holds a circular rack (*i.e.* a rod with teeth on it) into which the spur-wheels *d d* catch. From a pulley also revolving on the axle *e* these spur-wheels are driven by smaller spur-wheels *n n*. The belt connecting the pulley with another one fixed on the axle of the disc *e* is, as a rule, slack. When it is made tight the teeth wheels *d d* are caused to revolve, and set the boring apparatus in motion. For the purpose of changing the position of the cutting tool at the bottom of the shaft, a 'Krückel' (tiller) is fitted on to the boring rods at the surface, and is operated in

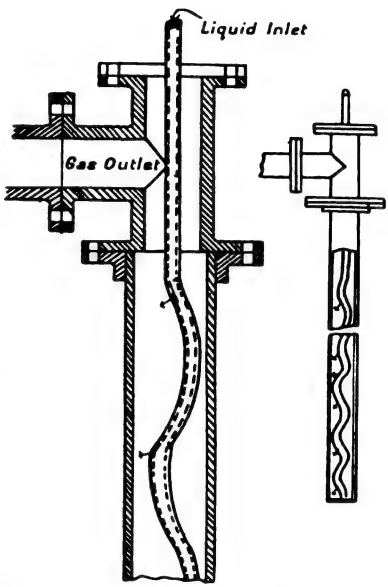
exactly the same way as when small bore-holes are being put down by the ordinary percussive method. For the lowering and taking out of the borer and pumps a steam crab is used, which is set up opposite the boring appliance, on the other side of the shaft scaffolding. The total weight of the tubular rods and the necessary pipes amounts to 297 lbs. per current metre (3·28 ft.). The cutting head is worked at the rate of fifty to sixty strokes per minute while boring goes on, the height to which the cutter is lifted being 18 to 20 cms. (7 to 8 in.). At every round the borer is set in afresh twenty to sixty times per minute according to the resistance offered by the ground. About fifty men are required in connection with this work. They are spread over three shifts of eight hours each. In every shift four men are employed on the boring stand.

In the deepening of Shaft IV. at the Rheinpreussen Colliery a sinking wall of 8·90 metres (29·19 ft.) clear diameter was built. The boring of the loose ground was then effected by means of a breaking appliance driven by hand labour. When the wall had been sunk through a layer of gravel of a depth of about 17 metres (56 ft.), the sinking working was temporarily suspended and the bottom of the shaft was filled up with concrete for a depth of 10 ft. After giving this concrete three months' time to become hard, a new sinking cylinder of 6·5 metres (21·32 ft.) diameter was built in and the sinking resumed, the percussive drill being used to penetrate through the concrete, which was cut at the rate of 4 ft. per day. This method of sinking has up till the present only been applied to loose water-bearing strata, and it has yet to be demonstrated that it would be equally successful in hard ground; but in view of the satisfactory results obtained in boring through the concrete layer, this does not appear to be out of the question.

Gobert's Freezing Method.—In the Poetsch freezing system, when any great depth is reached, the pressure of the liquid within the tubes becomes very high, and frequently brings about leakage of the liquid into the surrounding strata, which renders it impossible to freeze them effectually. In order to obviate this difficulty Gobert uses a cold transmitter, the pressure of which is lower than that of the water outside the tubes, while anhydrous ammonia vapour is used instead of the freezing liquid in Poetsch's system. With ammonia vapour very low pressures can be maintained, even at great depths, and if the tubes are not water-tight, instead of ammonia leaking out, the water from the surrounding strata would force its way in, and a coating of ice would be formed on the inside of the tubes, which would check the further inflow of water.

In order to vaporize the liquid ammonia in the tubes, these have to be connected with a suction and force pump. This pump sucks in the gas and compresses it into a liquid, with the help of a condenser, and then forces it into the freezing tubes. In order to avoid

the fall of the liquid to the bottom of the freezing tube, and to vaporize as much of it as possible in a given unit of time, the injector is made of a form spiral in one plane (see figs. 68, 69). The liquid, the entrance of which into the injector is carefully regulated, falls slowly in a thin stream within this spiral tube, and meets on its way a series of small orifices placed at various intervals in the tube. By these orifices the liquid escapes into the freezing tube, and vaporizes. In the Poetsch system the watery strata must be all frozen from the bottom upwards before the sinking can be proceeded with, but by Gobert's method the strata are frozen from the top downwards, thus allowing sinking operations to be started much sooner. Simultaneously with the sinking operations, fresh strata can be successively frozen, and so allow of continuous sinking. Freezing of the strata can be carried to great depths by this system; Gobert states that strata at a depth of 3000 ft. from the surface can be dealt with. For a recent sinking the cost of this system was £40 per foot.



Figs. 68 and 69.—Gobert's freezing tube.

Koch's Freezing System.—This system resembles that of Gobert's, but gaseous carbonic acid, ammonia, or a mixture of sulphur dioxide, is used as the refrigerating agent. Anhydrous ammonia, which has a density of 0·59, taking air as 1, and boils at $-40°$ C. at atmospheric pressure, is generally used. At the Washington Colliery, Durham, where the first two shafts sunk by this system in Britain were bored, the evolving brine used was a solution of 26 per cent. of magnesium chloride dissolved in hot water, which freezes at a temperature of $-34°$ C. The refrigerating agent is first subjected to a pressure of 150 lbs. per sq. in. by two compressors, and then delivered into a small receiver, from which it passes to the condensers, through a pipe 3 in. diameter, and thence into four tubes, each 1 in. diameter. These condensers are vertical iron cylinders, 10 ft. high and 5½ ft. in diameter, and contain, in tiers of four rings, 1600 ft. of tubing, 1 in. diameter, through which the ammonia circulates. About 4000 gallons of water per hour circulate through the condensers, the water being kept in constant motion by means of paddles. This cools the ammonia, reducing it to a liquid.

The condensers are connected to the refrigerators by piping 1 in. diameter, the refrigerators, like the condensers, being vertical iron cylinders, 10 ft. high and 7 ft. in diameter. These refrigerators, of which there are three, are jacketed first with 3 in. of peat-moss and then encased with wood. They are filled with the brine, and contain about 2000 ft. of tubing 1 in. diameter, through which the ammonia circulates after passing through reducing valves, which has the effect of reducing the pressure from 150 lbs. to about 15 lbs. per sq. in. At this point the ammonia is immediately changed from the liquid to the gaseous state, and as this can only be done by absorption of heat corresponding to the latent heat of vaporization, this heat is taken from the surrounding bath of brine, which is thereby greatly reduced in temperature.

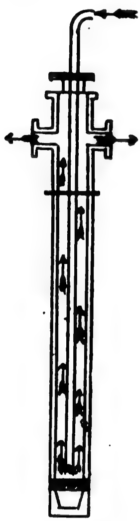


FIG. 70.—Freezing pipe.

At the Washington Colliery, before commencing the freezing process, the top of the shaft was enclosed, and the exposed pipes covered with straw. A hole was bored, and a pipe 18 ft. long inserted in the middle of the shaft, and the height and temperature of the water in the hole was noted as the gradual increase of ice-wall slowly caused the water to rise.

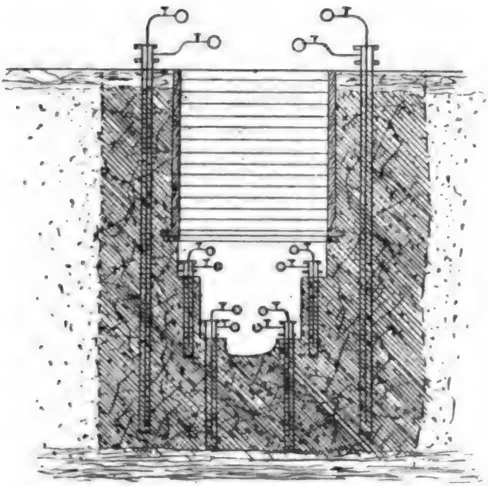
Poetsch's Method.—In this system of sinking, watery strata is artificially solidified by freezing. A series of bore-holes are first put down in the area where the shaft is to be sunk, and these are then lined with tubing through which a freezing solution of chloride of calcium is made to circulate by means of pumps. The freezing mixture, which is at a very low temperature, absorbs heat from the surrounding watery strata, which freeze into a solid mass, when the excavation of the shaft can be carried on in the ordinary way. Fig. 70 gives a sketch of the freezing pipes which are inserted into the bore-holes. They consist of an outer and inner tube, the freezing liquid being forced down the smaller inner tube circulating round the outer one, and escaping at the top, where it is led back to the refrigerating machine and used over again.

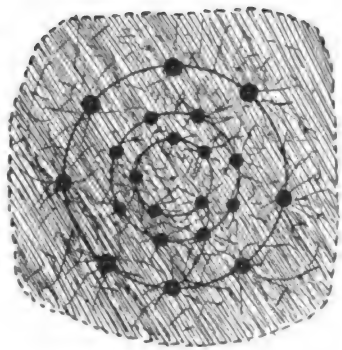
The large tubes, which are 6 in. to 8 in. diameter, are plugged up at the bottom with lead, cement, or any other substance that will render them water-tight; great care being taken to make this stopping secure, as the success of the operation practically depends on this precaution.

The number of tubes required will depend on the strata and the difficulty or otherwise of solidification. At a pit sunk at Lens in the north of France by this method, the area frozen was about 40 ft. diameter and 137¾ ft. deep, the number of tubes used was 28, and

the freezing of the strata took about 120 days. Figs. 71, 72 show the general arrangement of the tubes in the shaft.

In this system there is considerable risk of failure; for should there be any leakage or improper plugging of the tubes, the freezing mixture, which is itself uncongealable, may escape, and by permeating the strata render attempts to freeze the water futile. The freezing mixture or brine is usually a 20 per cent. solution of calcium chloride in water. It is cooled by means of ammonia, circulating in coils, at a pressure of 9 atmospheres (135 lbs. per sq. in.), in a liquid state. The temperature of the coils is 20 to 22° C. below zero, and





FIGS. 71 and 72.

the brine leaves the cistern at a temperature of about − 12° C. and returns to it at − 9° C.

Accessories to Shaft Sinking.—The operation of raising the excavated material during sinking is usually done by kibbles or buckets, which may be made either of iron or wood. The best form of sinking kibble is that in which the arms are fixed on trunnions with a catch at the top. Fig. 73 illustrates this kind of kibble. The great advantage of using one of this sort is that it can be completely and easily emptied without requiring to be detached from the winding rope, or even lowered on to a scaffold, for if it be swung clear of the shaft it can readily be emptied at any desired point by knocking up the catch *a*, which releases the arms and allows the body of the bucket to revolve on the trunnions *b b*. Sometimes kibbles constructed of wood and bound with iron are used instead of iron ones, but they are not so handy nor yet so durable as those made of iron. Fig. 74

shows a wood kibble which is well adapted for raising water and ordinary material.

Before the kibble is raised from the bottom of the pit, the sinker in charge ought to examine it to see that the fastenings are secure, and that no stones are likely to fall off, or that none are sticking to the outside of the kibble, liable to be knocked off during the ascent of the kibble, and possibly injure those who are working in the shaft. The same precautions should be adopted at the surface when the kibble is being lowered into the pit.

Safety Riders.—In the majority of shafts in process of sinking, the kibble is raised without being guided in any way; the common method being for the engineman to raise it a few feet from the pit-bottom when it is filled, and it is steadied for a few moments by one of the sinkers, and then drawn right away to the surface. This method acts very well if there is plenty of space in the shaft, and the depth not great; but when the depth becomes considerable, and cross-buntons require to be fixed, particularly in rectangular shafts, there is danger of the kibble catching these, and doing much damage to the sides of the shaft, and causing injury to the men at the pit-bottom. To obviate risks, guides are sometimes carried down as the sinking proceeds, and a rider employed to run between the conductors and guide the kibble, and also to keep it from swinging.

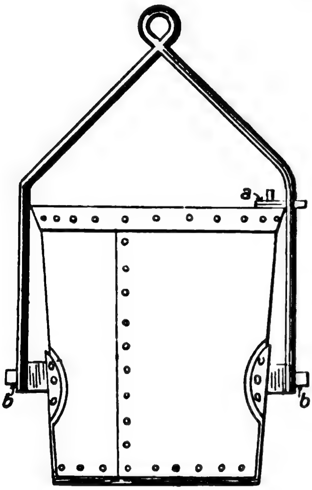
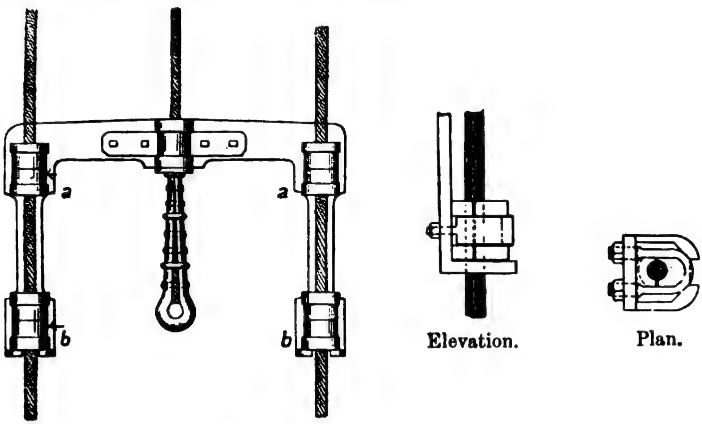


FIG. 73.—Iron kibble.

FIG. 74.—Wooden kibble.

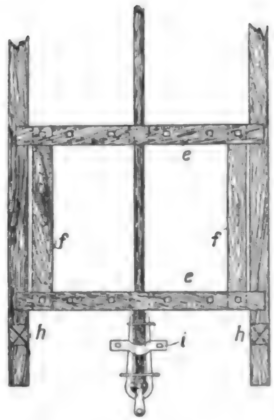
Fig. 75 shows the construction of a rider, made wholly of iron, to suit wire rope guides. Such a rider runs upon four bushes connected to the arms, the winding rope passing through an opening in the centre sufficiently large for the rope to pass through freely, but too small to permit the capping to do so. At a short distance from the pit-bottom conductors are fastened, or a projection is fixed to them, so that they may grip the rider when it reaches that point. The rope continues to descend through the central opening until the kibble reaches the pit-bottom, while the rider is securely held above. On the upward journey the rope runs through until the capping strikes the rider, which is then carried up to the surface, guiding the kibble during its ascent. Figs. 76, 77 show the details of the bush and gland which run on the rope at *a a* and *b b*.



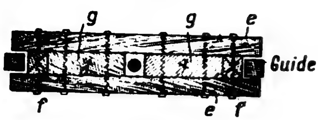
FIGS. 75, 76, and 77.—Iron rider.

When wood conductors are used a differently constructed rider is required. A form of conductor which is simple and efficient under these circumstances consists of four pieces of wood *e e* (figs. 78, 79), connected by two upright pieces *f f* firmly bolted together. The space between these is filled by pieces of wood *g g*, and only an opening about 4 in. square is left in the centre for the winding rope to pass through. Near the bottom of the guides two cleats are fixed, *h h*, for the rider to rest on, while the kibble proceeds to the pit-bottom. On the capping of the rope one or two pairs of glands *i* are fixed, for the purpose of catching the rider and carrying it to the surface during the ascent of the kibble. The advantages of using a rider during sinking are, that the winding of the kibble can be carried on at a much greater speed than if no rider be used while, as before stated, it is prevented from swinging about and so endangering the men working below.

The guides, if composed of wire ropes, should be frequently examined and kept well lubricated, particularly during frosty weather. Great care ought to be taken to keep ice from forming on the guides, as such obstructions prevent the rider from running freely, and it may then stick in the shaft and perhaps fall away and do much injury. Fatal accidents have occurred through the rider sticking and then dropping away suddenly.



Elevation.



Plan.

Figs. 78 and 79.—Rider for wood conductors.

Ventilating Shafts during Sinking.—While sinking is going on a sufficient supply of air must be provided at the bottom of the pit, to clear away the smoke due to blasting, and enable the men to work. This may be accomplished—

(1) By dividing the shaft by means of a close brattice and connecting one side of the engine chimney or stack.
(2) By carrying down a column of steam pipes and allowing steam to escape through a jet or nozzle in the closed compartment of the shaft.
(3) By ventilating the pit, by erecting either a temporary or the permanent fan and connecting it with the pit.

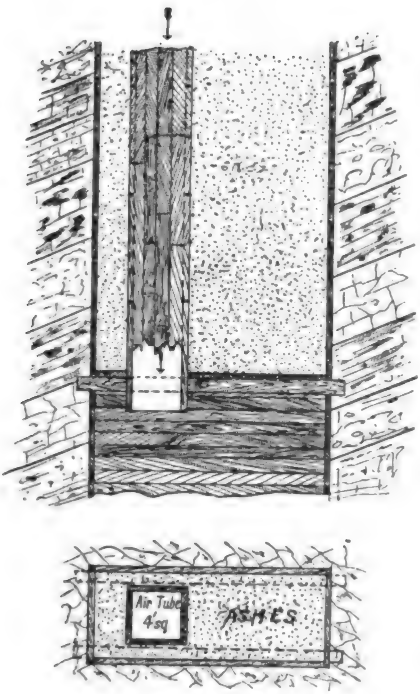
The first method is sometimes used, but, of course, would not be suitable if fire-damp were expected to be given off freely. Connecting the air drift to the flue of the chimney stack acts in the same way as a ventilating furnace underground by heating the air current, and thereby causing a circulation of air in the shaft.

The steam jet is a simple and handy way of ventilating shafts during sinking, and can be very easily applied, particularly if steam pipes have to be carried down to pumps in the shaft. Often the heat given off by these pipes is quite sufficient to ventilate the shaft without the aid of a steam jet.

Probably the best method is, however, to use a small temporary fan to force air down to the bottom of the pit. At Viewpark Colliery, Uddingston, while two shafts were being sunk, with a distance of about 50 ft. separating them, a small fan was used, connected to both shafts by a wooden drift or box 3 ft. high and 2 ft. broad, and made of flooring deals closely jointed together. Each shaft was divided by a close brattice and a connection made to the fan drift or air box. Smaller boxes were carried down the shaft. Sometimes pipes of large diameter made of thin sheet-iron are used

instead of boxes. A force fan is preferable to an exhaust fan for ventilating sinking shafts, as the bottom of the pit, after blasting, will be more quickly cleared, and the men can resume work sooner.

Enlarging Shafts.—Shafts sometimes become too small for the amount of work required to be done in them, and require to be enlarged. If winding has to be completely stopped and underground operations abandoned while such enlargement is taking place, the best method would be to entirely fill up the shaft with some light, loose material, and start the enlargement from the surface, and carry on sinking in the usual way. If the regular work of the colliery has, on the other hand, to be carried on, and the shaft contains pipes, etc., which it is undesirable to interfere with, then enlarging a shaft is not such an easy matter to accomplish. Each case must of course be dealt with according to the circumstances. The enlargement of a shaft of which the writer has personal knowledge was carried out in the following manner.



Figs. 80 and 81.—Timbering of shafts.

The colliery consists of two rectangular shafts, one being used as an up-cast and the other as a down-cast. The lining of the former showed signs of giving way, and the shaft had also departed from the vertical, while repairs of the wood lining had made it smaller than it was originally. It was determined to renew the whole of the lining, render the shaft vertical and enlarge it somewhat, while at the same time the whole of the winding was to be carried on at the down-cast.

To have filled up the pit completely would have stopped ventilation, and consequently stopped work by the colliers. The enlargement was therefore carried out in stages of 10 fms. or so, by putting in a scaffold, resting on strong beams, in the shaft, this scaffold being completely closed with the exception of an opening of about 4 ft. square, to allow of a wood box passing through for

the purposes of ventilation (see figs. 80, 81). This air-box, constructed of planking 9 in. × 3 in., firmly fitted together, was carried down past the scaffold for a short distance, and also up to the surface, and connected to the fan. The shaft was now filled in with ashes to the surface. The enlarging of this portion of the shaft was then proceeded with and new lining put in. The other sections of the shaft were dealt with in a similar manner until the whole shaft had been renewed to a depth of nearly 200 yds. The work was carried out expeditiously, and the whole of the output was dealt with at the other shaft. This method may be easily understood from figs. 80, 81.

Another method which the author has seen used for enlarging shafts is to use, instead of the wooden boxing described above, a wrought-iron or steel cylinder, 3 or 4 ft. diameter and 18 or 20 ft. long, an old flue of a Lancashire boiler serving well for the purpose. A strong scaffold is put in, leaving an opening for the tube to move through, and the pit is filled with ashes or other débris as already described. The tube is hung by a steel wire rope led from a steam winch on the surface, and as each section of enlargement is carried on, the tube is lowered so as to always keep the top of it a little distance above the filled-in débris. By this method a scaffold requires to be put in and the shaft enlarged in sections of 18 or 20 ft., or according to the length of the tube used. This system has the advantage that the iron tube is not so easily damaged as the wood boxing if blasting has to be resorted to.

CHAPTER IV.

EXPLOSIVES.

Definition.—An explosive is a substance the decomposition of which results in the sudden expansion of its components into a volume of heated gases many times exceeding its original bulk.

The strength of an explosive depends upon the volume of gases liberated, the rate at which decomposition proceeds, and the temperature of ignition. The gases liberated by the ignition of gunpowder, for instance, amount to about 2000 times the original volume of the powder used. The force exerted by ordinary blasting powder has been ascertained to be about 22,000 foot-pounds per sq. in.

The actual work performed by any explosive used in blasting operations is limited by incomplete combustion, compression, etc., by waste of energy in cracking and in heating material not displaced, and by the escape of gases through the shot-hole and through fissures in the rock.

The efficiency of explosives, *i.e.* the proportion borne by the work done to the theoretical energy liberated, has been estimated to range from 4 to 33 per cent.

Classification of Explosives.—Explosives may be classified in different ways, such as rending and shattering, or high and low, but the usual systems adopted are: (1) according to method of igniting; (2) according to composition. Under the first head they may be subdivided as follows:—

Explosives the decomposition of which is due to simple combustion, as in the case of ordinary gunpowder.

Explosives in which detonation occurs simultaneously throughout their mass, as Ammonite, Amvis, Bellite, Roburite, etc.

Explosives which partly detonate and partly burn, such as Carbonite, Kynite, Gelignite, etc.

Classification according to Composition (Cundhill):—

Gunpowder, ordinarily so-called.
Nitrate mixtures other than gunpowder.
Chlorate mixtures.
Nitro-compounds containing nitroglycerine, including the dynamite series.
Nitro-compounds not containing nitroglycerine, such as guncotton, etc.
Miscellaneous explosives.

Gunpowder is largely used. It is cheap, comparatively slow in action, and therefore suitable for coal and soft rocks, and less dangerous than some of the nitro-compounds. On the other hand, it is very dangerous in the presence of fire-damp and coal-dust, and its use is now prohibited in certain collieries by order of the Home Secretary.*

Gunpowder, if exploded in large quantities, is also dangerous to life, owing to the large percentage of carbon monoxide it gives off, and no explosives which give rise to this gas ought to be used for extensive blasting in mines, because of the risk of injury to health, and also because even small traces of carbon monoxide have been proved to render mixtures of coal-dust and air highly explosive, a point frequently overlooked in experiments with explosives. On firing 1¼ lbs. of blasting powder, over 3 cub. ft. of combustible gas, consisting chiefly of carbon monoxide, would be produced, and this, when mixed with pure air, would give over 10 cub. ft. of an explosive, or, at least, a rapidly burning mixture. The approximate composition of ordinary gunpowder is: Nitrate of potassium (saltpetre), 75 per cent.; carbon, 15 per cent.; sulphur, 10 per cent.

When gunpowder is exploded 56 per cent. of solid matter is formed and 44 per cent. of gas, or roughly, the solid matter is to the gaseous as 6 to 4. Ordinary blasting powder explodes at 600° F.

By the explosion of ordinary powder the following gases are produced †:—

	Volumes per cent.		Volumes per cent.
Carbon dioxide, . .	32·15	Sulphuretted hydrogen, .	7·10
Carbon monoxide, . .	33·75	Marsh gas, . . .	2·73
Nitrogen, . . .	19·03	Hydrogen, . . .	5·24
			100·00

From the foregoing it will be seen that gunpowder gives off a large percentage of carbon monoxide, which, as already stated, is very objectionable. The sulphur is also objectionable, and is by many makers reduced to a minimum. Gunpowder is very effective in breaking down coal, and is readily kept in good condition.

The following are typical blasting agents of the gunpowder type:—

	Special Bulldog Powder.	Bobbinite No. 1.
Nitrate of potassium, . . .	84–86	62–65
Charcoal, ...	12–13	17–19½
Moisture, ...	...	2½
Carbonate of magnesium, . .	2½–3½	...
Sulphur, ...	...	1½–2½
Sulphate of ammonium, . . } Sulphate of copper, . . . }	...	13–17

* See Coal Mines Explosives Orders, 1897–1904.

† *Ore and Stone Mining*, Prof. Le Neve Foster, sixth edition, p. 223.

To be compressed to a pellet, density 1·42, and, in the case of bobbinite, to be coated with paraffin wax melting at 120° F. Both explosives to be fired by electric fuse containing 5 grains of gunpowder, or with equivalent efficient explosive.

Other varieties of gunpowder, introduced of recent years, are the following :—

Constituents.	Argus Powder.	Earthquake Powder.
Nitrate of potassium, . . .	70–82	78–81
Carbon, ...	17–20	19–28
Distilled or pure sulphur, . .	½–1	⅛ (optional)
Oxalate of ammonium, . . .	...	...

Constituents.	Elephant Brand.	Oxalate Powder.
Nitrate of potassium, . . .	74–76	63–73
Carbon, ...	14½–15½	12–15½
Pure or distilled sulphur, . .	9–11	...
Oxalate of ammonium, . . .	...	13½–16½

Chlorate Mixtures.—Explosives containing chlorate of potash are regarded as too dangerous for mining purposes, being peculiarly sensitive to slight shocks, blows, etc.

Nitrate Mixtures other than Gunpowder.—In this class of explosives nitrate of sodium (Chili saltpetre) is substituted for potassium nitrate. Such mixtures are cheaper, but are absorbent, or deliquescent, *i.e.* they take up moisture from the atmosphere and are therefore unsuitable for mining purposes.

Nitro-compounds containing Nitroglycerine.—In this class are included all those 'high' explosives which are so useful in mining, and particularly in blasting operations in hard rock. Nitroglycerine is a light yellow, oily liquid, having a specific gravity of 1·6. It freezes at 40° F., and explodes with great violence at 360° F., or when subjected to a sudden shock. It is less sensitive to blows and detonation when frozen than when in the liquid state. Its use in the pure state is forbidden in Britain.

Blasting Gelatine.—This is one of the most powerful explosives used in mining. Its manufacture is both difficult and dangerous, but when once made it is one of the safest of explosives. It contains 93 per cent. to 95 per cent. of nitroglycerine, and 5 per cent. to 7 per cent. of nitro-cotton.

It is less rapid in detonation than dynamite, and is quite insoluble in water, in which it may be kept without deterioration. In its plastic state it is less sensitive to shocks or blows than dynamite, but when frozen it is more so. A rifle-bullet fired into a frozen mass of it causes an explosion, while no effect is produced by the same treatment in an unfrozen condition. Its relative sensibility to detonation compared with dynamite has been accurately ascertained, 0·8 grain of 'cap mixture' being required to explode a given charge of No. 1

dynamite, while the best blasting gelatine requires, for the same charge, 3 grains.

Relative efficiency of different explosives with same charge :—

Blasting gelatine (93 per cent. nitroglycerine and 7 per cent. nitro-cotton),	1000·00
Nitroglycerine,	907·14
No. 1 dynamite,	642·85
No. 2 dynamite,	378·57
Gunpowder (extra strong),	194·28

It will thus be seen that blasting gelatine is about three times more efficient than ordinary dynamite, and about fives times stronger than gunpowder.

Dynamite.—This explosive is manufactured by impregnating diatomaceous Kieselguhr, a spongy earth obtained from Germany, with nitroglycerine.

Its average composition is : Nitroglycerine, 75 per cent. ; kieselguhr, 25 per cent.

When in a proper condition dynamite is plastic, may be safely handled, and is very convenient for use as an explosive. Irregularly shaped holes are easily charged with it, and it does not explode at ordinary temperature either by spark or flame, but requires detonation. When dynamite cartridges are at a temperature below 32° F. they will only detonate with difficulty. When their temperature falls below 40° F. they are not in a safe condition, owing to their increased sensitiveness to shock. When in a frozen condition they should only be thawed by the warming-pans provided by the makers, and not heated in tin cans over fires or carried about in trouser pockets, as is too often done by miners.

Relative Efficiency of Gunpowder and Dynamite.

For Equal Weights.		For Equal Bulks.	
Gunpowder	=1·00	Gunpowder	=1·00
No. 1 dynamite	=3·75	No. 1 dynamite	=6·00
No. 2 dynamite	=2·00	No. 2 dynamite	=3·30

The use of dynamite results in economy of labour and tamping, loose sand being sufficient. It can be used in watery rock, and gives off but little smoke.

Safety or Flameless Explosives.—In all fiery or dusty mines where it is absolutely necessary to prevent flame issuing from a shot on explosion, one or other of the numerous safety explosives must be used.

The Home Secretary has it in his power to prevent the use of such explosives as he may deem unsafe for mines, and before any explosive can be considered safe for such mines it must be tested at a station provided by the Government for this purpose at Woolwich.

The following is a complete list of the names of explosives permitted by the Act which came into force 1st January 1898, and was revised in

Coal Mines Orders of the 20th December 1902, of the 24th April 1903, of the 5th September 1903, and of the 10th December 1903 :—

NAME OF EXPLOSIVE.

Albionite.	Clydite.	Nobel Carbonite.
Ammonal.	Coronite.	Normanite.
Ammonite	Dahmenite A.	Pit-ite.
Amvis.	Dragonite.	Roburite No. 3.
Aphosite.	Electronite.	Saxonite.
Arkite.	Faversham Powder.	Stow-ite.
Bellite No. 1.	Fracturite.	Thunderite.
Bellite No. 3.	Geloxite.	Victorite.
Bobbinite.	Haylite No. 1.	Virite.
Britonite.	Kynite.	Westfalite No. 1.
Cambrite.	Negro Powder	Westfalite No. 2.
Carbonite.	Noble Ardeer Powder.	

Other explosives are being tested from time to time and added to the list. It must be understood that the above list of permitted explosives does not form a guarantee by the Home Office that the explosives are safe under all conditions; it only signifies that those named have passed the Woolwich test, and the mine-owner is left to choose the explosive that he thinks may be safest to use under the conditions prevailing at any given colliery. To assist mine-owners, however, the Home Secretary issued a notice in October 1899, intimating an additional test to which explosives already upon the 'Permitted List' might be subjected. The proposed test was more severe than the original one, and explosives which passed it will be placed on a 'Special List.'

The composition of these permitted explosives is as follows :—

	Albionite.	Arkite.	Britonite.
Nitrate of potassium, ...	$8\frac{1}{2}$–$10\frac{1}{2}$	21–23	31–34
Nitroglycerine, ...	$80\frac{1}{2}$–83	51–54	25–27
Nitro-cotton, ...	5–7	3–4	...
Wood-meal, ...	2–3	6–8	39–43
Chalk, ...	$\frac{1}{2}$	$\frac{1}{2}$	...
Oxalate of ammonium, ...	...	14–16	...
Carbonate of sodium, ...	...	...	$\frac{1}{2}$

The wood-meal to contain not less than 5 nor more than 15 per cent. of moisture. The cartridges to be of non-waterproofed parchment paper, and fired with an electric detonator No. 6.

	Cambrite.	Carbonite.
Nitroglycerine, ...	25–27	25–27
Nitrate of barium, ...	$3\frac{1}{2}$–$4\frac{1}{2}$	30–36
Nitrate of potassium, ...	28–32	
Wood-meal, ...	39–42	39–42
Sulphuretted benzol, ...	$\frac{1}{2}$	$\frac{1}{2}$
Carbonates of sodium and calcium, .	$\frac{1}{2}$	$\frac{1}{2}$
Oxalate of ammonium, ...	...	...

Wood-meal: moisture not less than 10 nor more than 20 per cent. Non-waterproof wrappers of parchment paper. Detonator No. 6.

	Coronite.	
Nitroglycerine,	38	40
Nitro-cotton	1	1½
Nitrate of ammonium,	26	28
Nitrate of potassium,	3	5
Stearate of aluminium,	11	14
Rye flour,	8	11
Wood-meal,	2	4
Liquid hydrocarbon of the paraffin series,	2	4
Moisture,	...	2½

Wood-meal and rye-flour: not more than 15 and not less than 5 per cent. of moisture; hydrocarbon to have a flash-point not less than 200° F.; the stearate to be free from mineral acid. Waterproof wrapper. Detonator No 7.

	Clydite.	Dragonite.	Fracturite.
Nitroglycerine,	25–27	34–37	51½–53½
Nitrate of barium,	32–36	...	...
Wood-meal,	38½–41½	and charcoal 11–32½	5–7
Sulphuretted benzol,	½	...	...
Carbonate of sodium, Carbonate of calcium,	½	...	...
Oxalate of ammonium,	8	...	14–16
Nitro-cotton,	...	2–3	3–4
Nitrate of potassium,	...	43–46	21–25
Vaseline,	...	5–6	...

Non-waterproof parchment wrappers. Detonator No. 6. Wood-meals (first two), 5 to 15 per cent. of moisture; fracturite, 5 to 17 per cent.

	Geloxite.	Haylite No. 1.	Kynite.
Nitroglycerine,	54–57	25–27	25–27
Nitro-cotton,	4–5	½–1½	...
Potassium nitrate,	18–22	19–21	...
Wood-meal,	5–7	12–14	39–42
Ammonium oxalate,	13–15	10–12	...
Red ochre,	1	...	...
Nitrate of barium	...	19–21	30–36
Mineral jelly (acid free),	...	6–8	...
Chalk,	...	...	½

Wood-meal, 5 to 15 per cent. of moisture (except kynite, 10 to 20 per cent.). Non-waterproof wrappers. Detonator No. 6.

	Nobel Ardeer Powder.	
Nitroglycerine,	31	34
Kieselguhr,	11	14
Sulphate of magnesium,	47	51
Nitrate of potassium,	4	6
Carbonate of ammonium,	...	½
Carbonate of calcium,	...	½

Non-waterproof wrapper. Detonator No. 3.

	Nobel Carbonite.	Normanite.
Nitroglycerine,	25–27	32½–34½
Nitrate of potassium,	28–32	42½–46½
Nitrate of barium,	3½–4½	...
Wood-meal,	39–42	7–9
Sulphuretted benzol,	½	...
Carbonates of sodium and calcium,	½	...
Charcoal,	...	1–2
Ammonium oxalate,	...	10–12
Nitro-cotton,	...	1–2

Wood-meal, 10 to 20 per cent. of moisture. Non-waterproof wrappers. Detonator No. 6.

	Pit-ite.	Saxonite.	Stow-ite.
Nitroglycerine,	25–27	58–68	58–61
Nitrate of barium,	31–35	...	...
Wood-meal,	40–43	5–8½	6–7
Carbonates of sodium and calcium,	½	...	...
Nitrate of potassium,	...	21½–30½	18–20
Chalk,	...	½	...
Oxalate of ammonium,	...	9–27	11–13
Nitro-cotton,	...	31–5½	4½–5

Wood-meal, 5 to 15 per cent. moisture. Non-waterproof wrappers. Detonator No. 6.

	Victorite.	
Nitroglycerine,	25	27
Nitrate of barium,	32	36
Wood-meal,	38½	41½
Sulphuretted benzol,	...	½
Carbonate of sodium, } Carbonate of calcium, }	...	½

Wood-meal, 5 to 15 per cent. of moisture. Non-waterproof wrappers. Detonator No. 6.

Nitro-compounds not containing Nitroglycerine.—These have, as their base, nitrate of ammonium, mixed with other substances. The more important explosives of this class are :—

	Amvis.	Ammonal.	Ammonite.
Nitrate of ammonium,	88–91	93–97	87–89
Wood-meal,	4–6	..	...
Moisture,	½	1	½
Di-nitro-benzol, } Chlorinated naphthalene, }	4–6	...	...
Metallic ammonium,	...	4–6	...
Di-nitro-naphthalene,	...	...	11–13

Amvis.—Chlorine not to exceed 1 per cent. of finished explosive. Special wrappers are required for all these explosives; they are fired with No. 6 detonators.

	Aphosite.	
Nitrate of ammonium,	58	62
Nitrate of potassium,	28	31
Charcoal,	3½	4½
Wood-meal,	3½	4½
Sulphur,	2	3
Moisture,	...	1½

Special wrapper and fuses required.

	Bellite No. 1.	Bellite No. 2.	Dahmenite.
Nitrate of ammonium,	82–85	92–95	91½–93½
Di-nitro-benzol,	15–18	5–8	...
Moisture,	¾	¾	1
Naphthalene,	...	...	4–6½
Bichromate of potassium,	...	...	1½–2½

Special wrappers, and No. 7 detonator.

	Electronite.	Faversham Powder No. 1.	Faversham Powder No. 2.
Nitrate of ammonium,	71–75	84–86	87–93
Nitrate of barium,	18–20	...	...
Wood-meal, Starch,	7–10	...	...
Moisture,	½	½	1
Tri-nitro-toluol,	...	10–12	9–11
Chloride of ammonium,	...	1–2	...
Chloride of sodium,	...	2–3	...

Electronite.—Wood-meal to be charred. Lead waterproofed case; detonator No. 7. The Faversham powders are not specially indicated.

	Negro-Powder.	
Nitrate of ammonium,	86	90·0
Tri-nitro-toluol,	9	11·0
Graphite,	1	3·0
Colouring matter,	...	0·1
Moisture,	...	1·0

Special cases. Detonator No. 6.

	Roburite No. 3.	Thunderite.
Nitrate of ammonium,	86–89	91–93
Di-nitro-benzol,	9–13	...
Tri-nitro-benzol,	...	3–5
Chloro-naphthalene,	2	..
Flour,	...	3–5
Moisture,	½	1

Special wrappers. Detonators No. 6 and No. 8 respectively.

	Virite.	
Oxalate of ammonium,	9	12
Nitrate of ammonium,	35	40
Nitrate of potassium,	33	38
Sulphur,	4	5
Charcoal,	10½	12½
Moisture,	1	2

Special wrapper. Electric fuse containing 5 grains of gunpowder or equivalent explosive.

	Westfalite. No. 1.	Westfalite. No. 2.
Nitrate of ammonium,	94–96	90–92
Nitrate of potassium,	..	3–5
Resin,	4–6	4–6
Moisture,	½	½

Special wrappers and No. 7 detonator.

Detonators.—These are generally made of copper caps containing a small quantity of fulminate of mercury, or a mixture of fulminate with chlorate of potash, the proportion of the latter varying from 5 to 40 per cent.

The Home Office issued an order in 1897 regarding the standard charge for detonators, which is to consist of a mixture of fulminate of mercury 80 per cent., and chlorate of potash 20 per cent., or some other explosive mixture of the fulminate class of not less strength than the above. Different explosives require detonators of different strength to explode them, and the manufacturers of explosives generally recommend a certain class or strength of detonator for use in blasting the different explosives which they produce. These strengths are usually denoted by numbers, and the following are those commonly in use:—

	No. 1.	No. 2.	No. 3.	No. 4.	No. 5.	No. 6.	No. 6½.	No. 7.	No. 8.
Charge per 1000 in grammes	300	400	540	650	800	1000	1250	1500	2000
Individual charge in grains	4·6	6·2	8·3	10	12·3	15·4	19·2	23·1	30·9

For rending explosives—*e.g.*, some of the blasting powders, Nobel Ardeer powder, etc.—a detonator of the strength of No. 3 is commonly employed; for explosives of the nitroglycerine class No. 6 or No. 6½ is the most suitable. Miners and shot-firers should therefore be provided with the class of detonator most suitable for the explosive they are using. As the whole success and safety of shot-firing almost entirely depends on the detonators, it is of the utmost importance that not only the proper strength of detonator be used, but also that the detonators be of a good quality. A common source of annoyance and danger in using inferior or under-strength detonators is miss-fires, which should be avoided at all hazards. Miss-fires, however, may

occur with the best quality of detonators if they are not carefully handled and stored. Detonators should be kept in a dry place, and under no circumstances should they be placed in sawdust, as is sometimes done, for they absorb the moisture from it and soon become useless. They should also be bought in such quantities as will suffice for short periods, as they soon deteriorate if kept in store. In taking them into the mine, detonators ought to be carried in a securely locked case or bag separate from the other explosives.

Fixing the Detonator.—To fire a shot, a piece of fuse, of sufficient length, is taken, cut clean, and inserted into the open end of the detonator, which is then inserted into the cartridge of explosive, taking care to have the detonator so placed that the explosive covers it. After placing the detonator in position, it should be securely tied to the cartridge with a piece of string. When placing the charge in the shot-hole the detonator end of the cartridge should be towards the mouth of the hole, one detonator only being used for each charge.

Precautions against Fire-damp Explosions.—In order to avoid the danger of fire-damp explosions, arising from shot-firing, in fiery mines, strict observance should be paid to the following rules with regard to the detonator:—

(1) Explosives should always be exploded by a sufficiently powerful detonator. Newly-introduced explosives should be tested first to ascertain the strength of the detonator required.
(2) The fulminating portion of the detonator must be properly enclosed, for only such caps can be depended on as do not suffer from leakage.
(3) Detonators should be tested to see if they are in good condition.
(4) Wherever possible, detonators should be specially re-dried before being used.
(5) If explosion is to take place in wet or damp ground, the point of junction with the friction fuse should be well protected by some waterproof covering.*

Friction-detonators, *i.e.* detonators which are fixed to the fuse when manufactured, are safer during transport, less liable to be jarred in tamping, and safer to handle.

Firing the Charge.—Except where blasting is done by electricity, the charge is fired either by germans, squibs, or safety fuses, the latter being most largely employed in coal-mining. Germans or squibs consist of small cylinders of cardboard or stiff paper filled with gunpowder, and are inserted into the shot-hole. To the outer end a piece of cotton wick, saturated with oil, is attached. When the charge is ready for firing, the wick is lighted.

Safety Fuses.—Many safety fuses have been devised. They mostly consist of a fine column or central core of gunpowder, surrounded by flax, cotton, or similar materials. Taped fuses are protected by an external varnished coating, and are adapted for use in wet ground. In using such a fuse, the charge to be fired must be enclosed in a cartridge, into one end of which the fuse is introduced. The gutta-

* *Trans. Fed. Inst. Min. Engs.*, vol. x. pp. 550-557.

percha fuse is surrounded by a coating of that material, and has also an outside coating of waterproof varnished cloth, so as to preserve the guttapercha, as the latter becomes very brittle when exposed to air. Fuses should be kept or stored in a dry place, so that the core of powder may not be affected by damp, and care should be taken that they do not come in contact with greasy or oily matter, as this rapidly penetrates the outer covering to the gunpowder and prevents the proper burning of the fuse. Ordinary fuse burns at the rate of about 3 ft. per minute.

The disadvantages of using ordinary safety fuses are: (*a*) Uncertainty of burning speed of the fuse; (*b*) danger of miss-fires through defective fuse; (*c*) dangers of shots hanging fire; (*d*) ignition of explosive gases from 'spit' of safety fuses; (*e*) ignition of explosive gases from burning fuses; (*f*) dense smoke given off from the burning fuses.

Firing Shots by Electricity.—In many collieries, especially those in which safety lamps are used, or which are dry and dusty, it has now become the custom to fire the shots by electricity. There can be no doubt but that this system of firing shots, when properly carried out, is very much safer than blasting with the ordinary fuse and detonator, as it allows the workmen to retire to a place of safety before the charge is exploded, and there is much less danger from 'hang-fire' or 'miss-fire' shots, or from the premature explosion of the charge.

Electric Fuses.—In exploding charges of explosives by electricity, fuses, or detonators with wire attachment, are used. These fuses are generally of two kinds, viz., high- and low-tension fuses. High-tension fuses have their terminal wires bridged by a chemical paste or priming powder of relatively high electrical resistance. When a current is sent along the cables "the electrical energy at the fuse (wire) terminals is (owing to the insufficient conductivity of the bridging composition) converted into heat energy; the heat cannot dissipate with sufficient rapidity, therefore the temperature rises to the point of ignition of the bridge or priming. The latter bursts into flame and in turn fires the explosive compound at the end of the copper cap"; or, simply, the explosion is caused by the electric current heating the priming compound to ignition point. In the low-tension fuse the terminal wires are connected or 'bridged' by a short bridge of fine iridio-platinum wire (the wire is an alloy of 80 per cent. platinum and 20 per cent. iridium). The current in passing through this bridge raises its temperature sufficiently high (due to the same cause as in the high-tension fuse) to ignite the priming; the priming in turn, as above, fires the fulminate of mercury compound. In the Dortmund district of Germany, fuses called 'Spaltglühzündor,' occupying an intermediate position between high- and low-tension fuses, are used for firing single shots. They are essentially low-tension igniters, but the platinum bridge is replaced by a finely

divided conducting substance, such as graphite or coal-dust, mixed with the igniting substance, and the resistance opposed to the current is far greater than that of a small platinum wire, being generally about 5000 ohms. The quantity of current required to ignite the few particles of dust between the wire terminals is very slight, not exceeding at most $\frac{1}{100}$th of an ampère. Such igniters are said to combine the advantages of both the high- and low-tension fuses without their disadvantages.

Exploders.—For igniting or exploding the fuse, small electric machines or exploders are used, that most largely employed being known as a magneto-exploder. It consists essentially of an armature, revolving between the poles of a set of permanent electromagnets of hardened steel. The armature, which is wound to a high resistance, is made to revolve rapidly by means of a rotary crank connected to geared wheels in contact with the armature spindle. By this means an electric current of high potential is generated sufficient to explode the fuse. Afluidic or dry battery and secondary battery exploders are also used for the electric firing of shots in mines.

Firing Cables or Conducting Wires.—In order to allow a safe distance between the blasting charge and the shot-firer, a suitable length of cable, for conveying the current, must be employed to connect the exploder to the fuses. The length required will depend upon the nature of the blasting, *i.e.* whether it is in rock or coal, and the place where the shots have to be fired, but the minimum length should not be less than about 50 ft. for coal, and a longer length for stone-work in narrow drifts. The most suitable length, however, to be used in every colliery cannot be arbitrarily stated, but must be fixed from what is found to be best by practical experience.

Simultaneous Firing.—When blasting with electricity in shaft sinking or stone drifts, the shots are ignited simultaneously in order to obtain the maximum rending effect, although it is questionable if this effect is always got from such a method of blasting. Some persons, indeed, hold that independent firing gives much better results, for the reason that if all the shots are ignited at once it cannot be expected that one shot will help the other; but if the centre charges, say in a stone drift or at the bottom of a sinking pit, are fired first, so as to loosen the middle portion, the side charges should then operate under the most favourable conditions. To enable shots to be fired independently with electric blasting, a system has been brought into use by the adoption of a combination of electric ignition and tape-fuse. By this system a retarding action is got by inserting a piece of tape-fuse between the electric igniter and the detonator. For simultaneous firing two systems of connecting the fuses to the exploder are usually employed, known as the series and parallel systems. In the series system the line and fuse wires are coupled consecutively, one wire of the fuse being connected direct to the cable, the other wire connecting the first shot to the

second, the second to the third, and so on until all the charges are joined up, after which the remaining wire is coupled up to the second cable. Low-tension fuses are generally fired by this system.

With the parallel system the two firing cables are connected to the last charge, forming a parallel line, and then the wires of the other fuses are coupled up to them alternately.

Where electricity is not employed for blasting, simultaneous firing of a number of shots can be carried out by using the Bickford patent volley-firer in conjunction with the same maker's instantaneous fuse. This appliance consists of a little instrument devised so as to contain an ordinary safety fuse at one end and at the other a set of instantaneous fuses, the set varying according to the number of charges to be blasted. Between the end of the safety fuse and the ends of the instantaneous fuses is inserted an explosive disc, the action of which is such that, on the communication of fire from the safety fuse, the whole of the instantaneous fuses are immediately ignited, the latter burning at the rate of about 100 ft. per second, giving practically the same result as with electric firing. Any number of shots up to sixteen can be fired simultaneously by this apparatus.

Igniting Shots in Fiery Mines.—In fiery mines or mines where safety lamps are exclusively used, and where blasting has to be done, the ignition of the shots may be carried out by (*a*) safety lamp; (*b*) special contrivances like Bickford's patent fuse lighter; (*c*) by electric firing. Electric blasting has already been fully dealt with.

Blown-out Shots.—These may be a source of great danger in mines which are dry and dusty and where fire-damp is given off. They ought to be carefully guarded against in blasting. Blown-out shots are brought about chiefly by (*a*) insufficiently and badly tamped holes; (*b*) an insufficient charge of explosive for the work to be done; (*c*) the shot-holes being drilled beyond the line of holing or kirving. The charge should be well, but not excessively, tamped with good surface clay or fireclay, free from stones or hard nodules, the tamping being firmly rammed back with a wooden stemmer. The hole should not be drilled beyond a point 6 to 8 in. from the back of the holing, and in stone work especially, where the holing has often to be blasted out, the holes should be drilled at a suitable angle and length according to the kind of explosive used.

Position of Shot-holes.—The most suitable position for the charge depends upon various circumstances, a proper knowledge of which can only be obtained by actual practical experience. All 'joints,' 'backs,' 'lypes,' 'partings,' etc., must be carefully avoided, and the position of the hole for the charge so placed that the resistance in every direction may be as nearly equal as possible from the expected plane of fracture.

As an explosion takes effect along the line of least resistance, if there are any joints or cracks near the hole, they will determine the direction of fracture, and the charge will have comparatively

little effect in any other direction. In the case of a sump-hole, for instance, in a sinking pit, the line of least resistance will be the shot-hole itself, and in such a case a heavier charge of gunpowder than ordinary must be used, and it will have to be well stemmed, or a strong explosive occupying little bulk must be employed, such as dynamite.

Prevention of Flame Communication.—*The Water-cartridge.*—In the presence of fire-damp, what is known as the 'water-cartridge' is used, in conjunction with dynamite or other explosives. The water-cartridge consists of a cylindrical case of specially prepared waterproof paper 18 in. long and 2 in. diameter. In the centre of this case is placed the explosive, kept in position by thin metallic webs, the end of the cartridge having the detonator for electric firing fixed to it by wire. The space between the charge and the paper cylinder is filled with water, and the outer end firmly tied round the projecting wires. The water-cartridge is most largely used for dynamite blasting, but it cannot be said to give very satisfactory results, and has only been used to a very limited extent. The difficulty hitherto experienced with the water-cartridge is that the water is so liable to escape and the blast takes place in an empty cartridge. Instead of using water alone, a mixture of soap and water has been tried and found more reliable and effective. The water and a certain proportion of soap are first boiled together, and the resulting viscous liquid is then filled into an india-rubber bag or cartridge, and used in much the same way as the water-cartridge. Any other cheap gelatinous compound may be used for the same purpose.

Wet Sand.—Sometimes common sand, moistened with water, is used; the cartridge containing the charge being placed in the centre of a paper covering made large enough to admit of ½ in. of sand being placed all round it. The paper cartridge should be made thoroughly water-tight by being soaked in oil or grease and then allowed to dry.

Wet Moss.—Stemming the charge with wet moss is another means employed to prevent flame being communicated to the surrounding atmosphere, and is said to be effective when used with gelatine and dynamite. The same result may be obtained by using a stemming of moist clay, and probably this is as effective as anything which may be used.

Cost of Blasting.—This will vary greatly according to the kind of explosive used, the system of firing adopted, and the hardness of the coal or strata the shots are fired in. The cost for blasting will vary from about 0·35d. to 1d. per ton, or an average of about 0·7d. per ton in coal. Regarding the cost of firing shots in gaseous mines—exclusive of explosive and labour—by different systems, *i.e.* firing by low and high-tension electric safety fuses, and by the Bickford patent lighters and safety fuses *—Mr Frank W. T. Brain, in his evidence before the

* See *Report of Committee*, p. 488.

Departmental Committee on the use of electricity, gave some figures which are instructive, and are here reproduced :—

I. *Cost of firing* 1000 *Shots when using High-Tension Electric Detonators.*

	£	s.	d.
1000 4 ft. wires, electric high-tension, No. 6 detonators, complete, ...	£5	0	0
50 yds. firing cable, costing 10s., used, say, for 2000 shots, ...	0	5	0
Magneto exploder, costing 35s., plus repairs 10s., used, say, for 30 shots per day, two years, . . .	0	2	6
Total, . .	£5	7	6

Cost per shot = 1·29d.

II. *Cost of firing* 1000 *Shots when using Low-Tension Electric Detonators.*

	£	s.	d.
1000 4 ft. wires, electric low tension, No. 6 detonators, complete, ...	£5	16	0
50 yds. firing cable, costing 10s., used, say, for 2000 shots, ...	0	5	0
Magneto exploder, costing 35s., plus repairs 10s., used, say, for 30 shots per day, two years, . . .	0	2	6
Total, . .	£6	3	6

Cost per shot = 1·48d.

III. *Cost of firing* 1000 *Shots when using Bickford's Patent Safety Fuse and Igniters.*

	£	s.	d.
4000 ft. Bickford's fuse, ...	£4	3	4
1000 Bickford's igniters, ...	3	2	6
1000 No. 6 detonators, ...	1	15	0
Total, . .	£9	0	10

Cost per shot = 2·17d.

From these figures it will be seen that the high-tension system is 0·19d. per shot, or approximately 13 per cent. cheaper than the low-tension system, and firing by the Bickford fuse and igniters shows a difference of 0·88d. and 0·69d. per shot respectively for high and low-tension fuses, or a difference in favour of electricity of 68 per cent. and 46 per cent. respectively, for high and low-tension fuses. It must be distinctly noticed that these figures are a mere comparison of the cost of material alone.

On the whole, firing by electricity is cheaper. Regarding the total cost of blasting, *i.e.* including explosives, fuses and labour, it will vary greatly according to the kind of explosive used, the system of firing

adopted, and whether the colliery is a non-fiery one or a fiery, dry and dusty one. Naturally in the latter class of mine the cost will be somewhat higher than for non-fiery mines where open lights are used. The average cost for blasting with gunpowder is about 0·6d. per ton, and for safety explosives 0·9d. per ton of coal got, or a difference of 0·3d. per ton in favour of gunpowder, so that the cost for blasting is increased about fifty per cent. when safety explosives are used. With gunpowder the percentage of round coal got, in a number of experiments, averaged 62·2 per cent. and for safety explosives 62·0 per cent., so that so far as this is concerned there is not much difference between gunpowder and some of the safety explosives. In blasting rock there is, however, an estimated gain of 25 per cent. in using the latter.*

It may be taken for granted that many of the safety or permitted explosives are 50 to 100 per cent. dearer than blasting powder, but of course, on the other hand, a much smaller charge of such explosive will be required to do the work than if gunpowder was used.

* Paper by Henry Hall, H.I.M.

CHAPTER V.

MECHANICAL WEDGES, ROCK DRILLS, AND COAL-CUTTING MACHINES.

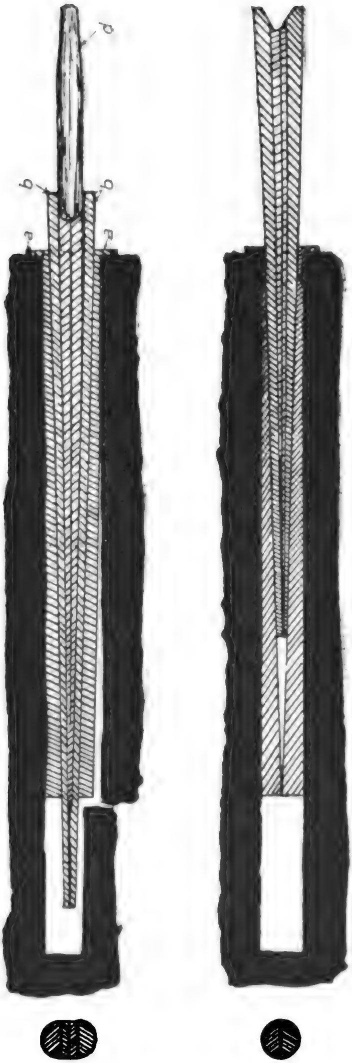
Mechanical Wedges in Coal-mining.—In underground excavations the coal seam or rock can very rarely be removed by the aid of picks alone, unless in very soft strata. In ordinary seams wedges are used to assist in bringing down the coal after it has been 'holed'—the commonest form employed being known as the 'feather-shaped' wedge. This wedge is also an adjunct to blasting in many mines; the coal being first loosened by explosives and afterwards wedged down.

Elliott Multiple Wedge.—In mines where blasting is prohibited, some mechanical method must be adopted for bringing down the coal or rock without the aid of explosives. The Elliott wedge is designed for such a purpose, and may be said to be an adaptation of the old plug and feather. The construction and use of the wedge will be understood from figs. 82, 83. To use the wedge a hole must first be bored out deep enough to hold the wedges. Into this hole are then inserted two portions of the wedge *a a*, tapered in front and increasing in thickness towards the further end. These pieces are constructed with the front portion turned back so as to grip the hole and prevent them from being driven out of position. Two other long-tapered wedges *b b* are now driven into the hole, and if these fail to bring down the coal, a third wedge, *d*, may be driven in between them and thus exert further pressure.

The advantages claimed for this wedge are that only a small hole requires to be bored, that the expansive force developed is great, the weight of wedges, etc., is small, and the first cost low. The wedges are made in two sizes: for holes 1¾ in. diameter, 2 ft. 6 in. long; and for holes 2 in. diameter, 3 ft. long.

Burnett's Roller Wedge.—In using the multiple wedge a great amount of power is necessarily lost in overcoming the friction of the parts sliding over each other, and the Burnett roller wedge has been designed to obviate this loss. In this appliance, the wedges, instead of being in sliding contact with the feathers or cheeks, are in rolling contact only; and as the rollers are arranged on each side of the

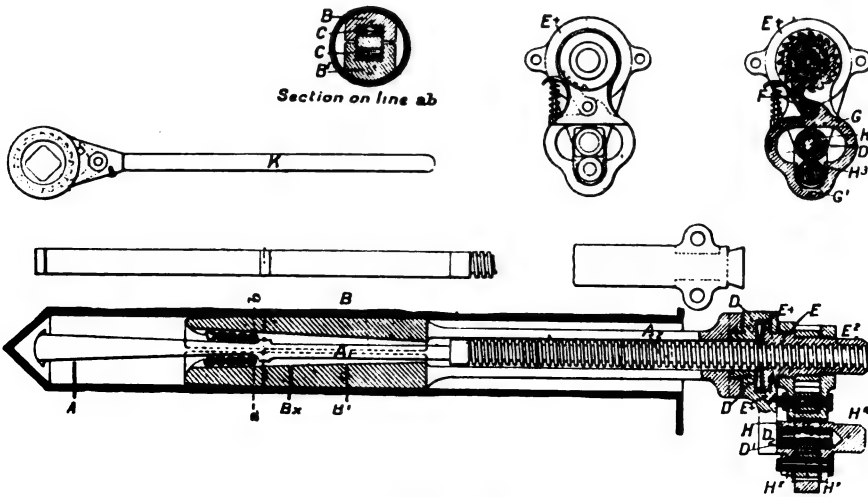
wedge, it will be understood that the latter travels twice the distance covered by the rollers on the stationary cheeks or feathers—the centres of the rollers having a velocity ratio equal to only half that of the advancing wedge. The power is thus doubled and the friction greatly reduced. The construction of the wedge will be seen from figs. 84–89. A is the wedge-bar formed in continuation of the plane A^1 and of the screw A^2. The plain part, A^1, is enclosed between two bars or feathers B B^1, which are placed in the bore-hole as shown. These bars are formed with a taper corresponding to that of the wedge, and the bottom feather Bx, which has to bear the greater strain, is made stronger than the top feather B. The rollers C C are placed near the end of these feathers B B^1. A nut E, placed on the screw, affords facility for pulling the inner wedge-bar A out towards the face of the coal, whereby the rollers C C, while ascending on the inclines of the inner taper bar A and the inclines of the outer taper bars B and B^1, force the two latter apart to the required extent. The nut E is formed with a flange, which is held in position by means of a bearing plate Ex, having lugs as shown. On the nut E, at its outer end, is formed a square, E^2, on which is fixed a ratchet lever.



FIGS. 82 and 83.—Elliott multiple wedge.

Application of Wedge.—The necessary holing having been completed and the coal drilled, the wedge is inserted. The ratchet lever K is then applied to E^2, which, in working, rotates the nut E, which, bearing against the shoulder D, begins to draw

out the wedge A. The operator continues this direct action of the handle until more power is required than can be thus directly obtained. The handle K is then shifted on to the square H^4 of the lever H, and by imparting successive oscillations to the same, the roller H^3, acting against the cam slot G^1 of the fork G, will cause the pawl (fig. 87) F to pass a tooth at each oscillation of the lever, and thus power in direct proportion to the ratio of the leverage between H and G will be developed.



FIGS. 84, 85, 86, 87, 88, and 89.—Burnett's roller wedge.

Hydraulic Wedges.—Water at high pressure has been used as a means of blasting, but, except with the Tonge hydraulic cartridge, the success of hydraulic blasting has been questionable.

Tonge Hydraulic Cartridge.—This cartridge has been introduced into a number of collieries in the Lancashire coal-field, and has given such satisfactory results that in one or two mines it has entirely superseded blasting by explosives.

The cartridge consists of a cylinder of steel 20 in. long by 3 in. in diameter, and having eight small duplex rams fixed radially along it. By a suitable arrangement of passages, a communication is made between each of these rams whereby simultaneous action can be obtained. By an ingenious contrivance a greater traverse is obtained by these rams than the diameter of the cylinder. This traverse is essential for the complete forcing down of the coal. Thin liners are used to prevent the rams cutting into the coal.

The cartridge is operated by a hydraulic pump, to which it is connected by a pipe. The pump is of special design, is mounted on

an adjustable stand, and fitted with water tank. The water required for the whole of the operation is about one and a half pints, but most of this returns to the tank at its completion, and can be used again. At the commencement of the operation the small handle is used, and when pressure has increased, an extension handle is slipped over it, and greater power is thus exerted. A pressure of 3 tons per square inch can be reached, and this represents a total pressure on the coal of over sixty tons. This is found adequate in ordinary seams, and the standard sizes are made for this duty. Special cartridges are made for extraordinary conditions.

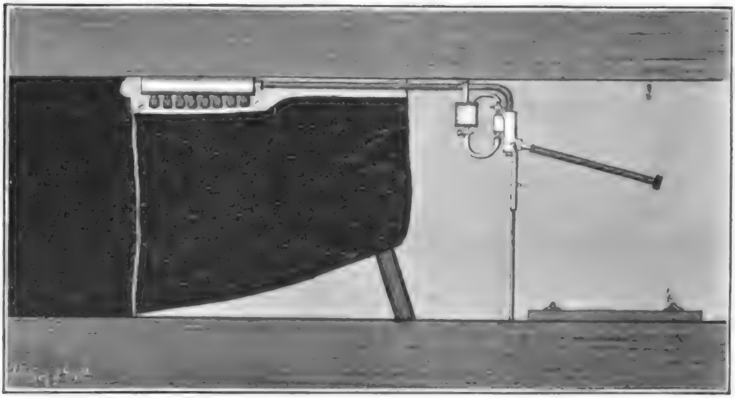


FIG. 90.—Tonge hydraulic cartridge, showing rams extended after use.

After the coal has been undercut, either by hand or coal-cutter, a hole 3¼ in. diameter is drilled about 3 or 4 ft. deep into the coal (slightly less than depth of holing). This is done by means of ordinary machine and spiral drills. The hole is put in parallel with the roof, and as near as possible along the parting to which the coal ordinarily comes off, or just below it. It is then cleared out. The cartridge, with one or more liners, and having pipe and pump attached, is pushed to the back of the hole, and the pump is left in position for the attachment thereto of the stand, which is adjusted to the required height. The water tank is filled, and hung on the pipe, the rubber suction pipe coupled, and the taps turned. The coal sprags are all left in tight. The pressure being fully on, the coal is heard to be rumbling and cracking. This is allowed to continue until the back portion of the coal is broken off, after which the sprags are slightly slackened. By a continuance of the pumping, the pressure is brought to bear at the front of the faces, and continues to spread until the operation is completed, when the

sprags are then withdrawn. The whole operation occupies about ten minutes. This system naturally secures absolute immunity from explosion, miss-fires, poisonous fumes, etc., while it is claimed that the coal is not so shattered. It has been in use three years at the Atherton Collieries, 19,000 shots having been made in one seam in a year, producing 40,000 tons from a 3 ft. seam.

Rock Drills.—Rock drills may be divided into two classes, viz.—(1) hand drills, (2) machine drills. In the first class the work is performed by manual power alone, while in the second class other methods are employed, such as hydraulic pressure or compressed air.

Hand Drills.— The commonest type of this class is that known as the ordinary 'ratchet' boring machine, which is now so extensively

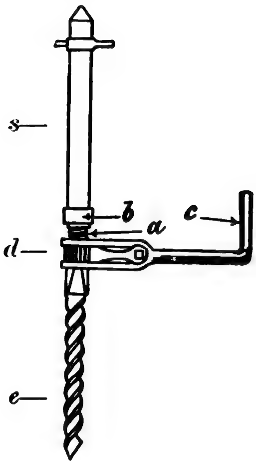


FIG. 91.—Hand drill.

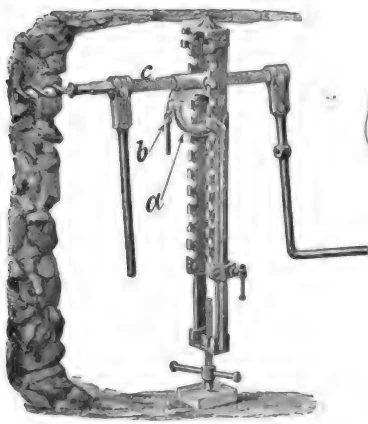


FIG. 92.—'Conqueror' machine.

used in all kinds of mining. It consists of a screw-spindle *a* (fig. 91), terminating in a hollow square, into which the drill *e* can be inserted, and working through a screw-nut collar *b*, which is fixed to, or composes part of, a hollow sheath *s*, for the screw-spindle to work in. When a hole requires to be bored, a prop is set firmly up, and the drill fixed against it with the boring-bit *e* inserted into a small hole or cut made with the pick in coal or rock. The handle *c*, which works on the ratchet *d*, is then turned and bores out the hole by a grinding action. When the full length of the drill is out, it is again worked back into the hollow sheath and a longer drill substituted, the same process being repeated until the hole is bored to the required depth. These machines are very handy, and will penetrate very hard stone. In the 'ratchet' machine a prop has to be used to

support the machine, and where hard rock has to be bored it is often difficult to keep the drill in position. To overcome this difficulty and save loss of time many machines are now provided with adjustable stands. In the 'Conqueror' boring machine such a stand is provided, and the screwed nut and hollow sheath is done away with, the screw-spindle actuating the drill working through a long screw thimble *a* (fig. 92), which can be fitted on any part of the stand to suit the height of working and position of hole. An iron sole-piece is used on which the machine may stand, the height of which can be varied by the adjusting screw, which at the same time tightens it between the roof and floor. There is also an adjusting sliding piece *b* for regulating the position of the boring bit. The drill can be worked either with one or two handles, according to the hardness of the material and the power required.

Machine-power Drills.—This class of machine has proved of great service in certain kinds of work. These drills are not so largely employed in coal-mining as in tunnelling, metal-mining, and quarrying, but there are often cases, such as the driving of mines in hard rock, sinking shafts through igneous rocks, etc., where they can be used with great advantage as auxiliaries in coal-mining. Possibly the fact that they require to be supplied with motive power such as steam or compressed air, the use of which involves considerable first cost in plant, and that skilled men are required to work them properly, will be against their extensive use in small coal-mines. There are, however, many collieries where compressed air is used for numerous purposes, amongst others for driving coal-cutting machines, and, in such circumstances, machine drills might be advantageously used.

In order to obtain the greatest benefit from the use of rock drills in sinking or tunnelling, it is necessary that a given length be driven by each series of holes, and that all the holes of each series should be fired simultaneously, either by electricity or by a quick-burning fuse. The special point in favour of the use of rock drills in hard stone is that the same work is effected in about half the time that it could be done by ordinary methods. The following average weekly results have been obtained in sinking shafts by the aid of rock drills, the rate of wages being that prevailing in 1894.*

Shaft 18 ft. diameter.	Depth sunk	Cost per Yard. Wages.			Explosives.			Total cost per yard.		
	Feet.	£	s.	d.	£	s.	d.	£	s.	d.
Very hard limestone with partings, ...	30	10	4	9	1	18	0	12	2	9
Coal measures, shales, and sandstones, . . .	39	7	12	6	0	19	0	8	11	6

* *Trans. Inst. Min. Engs.*, vol. viii. pp. 18–19.

The depth sunk by each series of holes was :—

Hard limestone,	4½ feet in	18	hours.
Hard sandstone,	5½ ,,	18	,,
Shale and sandstone,	6½ ,,	16	,,

The weight of stone lifted with each round of shots averaged, for hard stone 130 tons, and for moderately hard stone 150 tons.

These figures are significant, as the cost of sinking would have been at least 25 to 30 per cent. higher if the ordinary methods had been adopted, while the rate of progress is very much greater, which is an important factor in all sinkings. Of course there is the additional cost of plant and consumption of fuel to be taken into account. In the case cited, where four drills were used, an air-compressor having a cylinder 16 in. diameter by 24 in. stroke, with a 20 nominal horse-power boiler attached, was required. Each drill when working consumed about 30 lbs. of coal per hour, or taking the working time of the drills at four hours per day, the consumption of four drills would be about 480 lbs.

There are so many different kinds of machine drills in use that it would take up too much space to attempt to describe them all, and all that will here be attempted is to instance a few of the best known. Before doing so, it may be pointed out that the requirements of a good rock drill, as concisely stated by André * in his work on coal-mining, are as follows :—

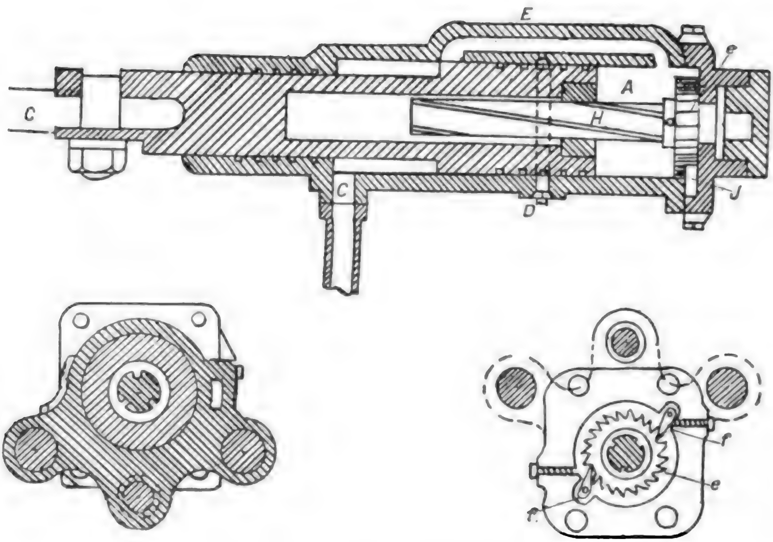
It should be simple in construction and strong in every part.
It should consist of few parts, especially of few moving parts.
It should be as light in weight as is consistent with the first condition.
It should occupy but little space.
The striking part should be relatively heavy, and should strike the rock direct.
No other part than the piston should be exposed to violent shocks.
The piston should be capable of working with a variable length of stroke.
Sudden removal of the resistance should not cause it any injury.
The rotary motion of the drill should take place automatically.
The feed, if automatic, should be regulated by the advance of the piston as the cutting advances.
The machine should be capable of working with a moderate degree of pressure.
It should be capable of being readily taken to pieces.

Darlington Drill.—This is one of the simplest and most effective machines in use. It consists essentially of two parts: the cylinder A (fig. 93), with its cover, and the piston with its rod, to which is attached the boring tool *c*. To give the latter a rotary motion, there is a spiral or rifled bar H, having three grooves, and being fitted at its head with a ratchet-wheel *e*, recessed into the cover of the cylinder.

Two detents *f f* (fig. 95) are also recessed into the cover, and are

* *A Practical Treatise on Coal-Mining*, by G. G. André, p. 148.

made to fall into the teeth of the ratchet-wheel by spiral springs. This arrangement of the wheel and the detents allows the spiral bar H to turn freely in one direction, while it prevents it from turning in the contrary direction. The spiral bar drops into a long recess in the piston, which is fitted with a steel nut, made to accurately fit the grooves of the spiral. Hence the piston during its instroke is forced to turn upon the bar, but during its outstroke it turns the bar, the latter being free to move in the direction in which the straight outstroke of the piston tends to rotate it. Thus the piston, with the boring tool, assumes a new position after each stroke. The total length of the Darlington drill is 3 ft., and the weight 100 lbs.



FIGS. 93, 94, and 95.—Darlington drill.

Adelaide Drill.—The Adelaide drill somewhat resembles the Darlington machine just described, but in it the air is used expansively, and it has only one moving part. The piston *c* (figs. 96, 97) works in a cylinder, having ports and passages so arranged that the air or steam is admitted and cut off automatically by the piston itself. Air enters through an annular port A, by which means the pressure is equalised on all sides of the piston-rod and unequal wear is avoided.*

The exhaust takes place through the port B. The piston itself is made to perform the action of a valve in the following manner: as soon as it reaches the port B, free communication is opened with the

* *Journal Brit. Soc. Min. Students*, vol. xiii. pp. 70, 71.

atmosphere and exhaust takes place, not only here, but also through the ports B_1, which have by this time passed outside the cylinder cover. The inlet aperture A being always in free connection with the air receiver, the pressure now acts on the small area at the front of the piston, and drives it backward, until this part is also

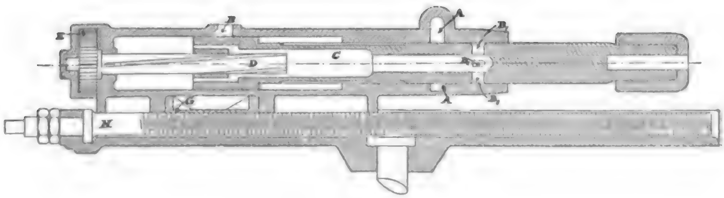


Fig. 96.—The Adelaide drill.

brought into connection with the exhaust; at the same moment, the ports B_1 come opposite the inlet A, compressed air enters through the hollow rod C, passes into the back end of the piston, and drives the drill rapidly forward against the rock. Admission takes place during half the stroke, the air working expansively during the second half.

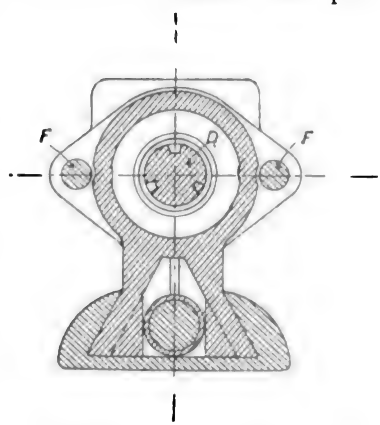


Fig. 97.—Section of Adelaide drill.

To rotate the drill there is a spiral or rifled bar D, having three grooves. It is fitted at its head with a ratchet-wheel E, which is recessed in the cover of the cylinder. Two detents or cams, also fixed into the cylinder cover, are forced by small springs to engage with the teeth on the wheel E. The feed is obtained through the rotation of the screw H in the nut G. As the admission both below and above the piston takes place some time before it arrives at the end of the stroke, a cushion is formed, and the piston is thus prevented from striking the covers.

Ingersoll Drill.—Fig. 98 shows the construction of the Ingersoll eclipse drill which is so largely used in American mines. Either steam or compressed air may be used as the motive power. The machine consists of the cylinder A, with the piston M, and the piston-rod B. Steam or air is admitted by a single spool valve C, and enters the valve chest at *o*, and when the piston has reached the end of its back stroke it passes round the valve to the passage N′, and then enters the port P′, and ultimately reaches the back end of the

cylinder. When the valve is reversed, the air goes past and enters the port P, to the front end of the cylinder. In the figure the port P is shown communicating with the exhaust E.

In the piston M are two recesses *s s*, which practically makes two short pistons out of one long one. There are two ports, F F′, leading to the exhaust, and two small ports, D D′, communicating cross-wise with the ends of the valve chest, *i.e.* the port D is connected with

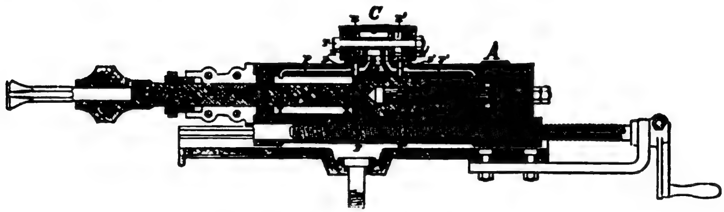


Fig. 98.—The Ingersoll drill.

the end R′ and D′ with R. At each end of the cylinder are placed two elastic cushions, H H′, to protect the cylinder covers from injury should the piston be suddenly relieved from resistance or the attendant fail to feed forward his machine as the drill advances. The drill is rotated by a rifled bar and ratchet-wheel. This drill can be had with cylinders varying in size from $1\frac{1}{4}$ in. diameter, which is used for light work, and holes 3 to 4 ft. deep and $1\frac{1}{4}$ in. diameter up to 5 in. diameter—the largest size made—which is used for holes 25 to 50 ft. deep and 3 in. to 6 in. diameter.

CHAPTER VI.

COAL-CUTTING BY MACHINERY.

Coal-cutting by machinery was introduced practically into British mines over thirty years ago, at a time when there was a great scarcity of workmen, and wages were high in the mining districts. As long ago as 1869 coal-cutting machines, of the Winstanley type, were being operated at collieries in the Wigan district, and about the same time, or shortly thereafter, Messrs William Baird, Limited, the well-known Scottish coal and ironmasters, introduced a coal-cutting machine of their own design to undercut the coal at their Gartsherrie Collieries, near Coatbridge. Since that time mechanical cutters have been used to some extent in the different mining districts, but until recently coal-cutting by machines has had only a limited application in British mines. Within the last few years, however, there has been a great increase in the number of machines in our mines, and in the near future coal-cutting by machinery will have a widely extended application all over the coal-fields of Great Britain.

In American mines during the last decade coal-cutting machines have been extensively employed, and at the present time over one-fourth, or 25 per cent., of the total output on that continent is cut by machines, while in Great Britain the quantity of coal got by machines is under two per cent. Some of the chief difficulties which have retarded the more extensive employment of coal-cutting machines in Britain until recently, have been badly designed machines, inferior workmanship, and the use of inferior material. This has to a great extent been largely remedied, and there are now some excellent machines at the disposal of colliery managers, suited to work under almost any ordinary conditions. Another difficulty which has had to be contended with has been the labour question, the workmen in many districts having a prejudice against the use of coal-cutting machines, on the ground that the introduction of such machinery will dispense with the need of hand labour. There has also been the difficulty of getting a sufficient supply of skilled men to operate and supervise the machines, and consequently in many cases incompetent men are operating machines with unsatisfactory results. This difficulty will lessen with the increase of skilled operators.

The following table shows the number of coal-cutting machines at work in the various mining districts in the United Kingdom for 1902, and the different types of mechanical cutters used, as given by Sir C. Le Neve Foster, in his Annual General Report and Statistics of Mines and Quarries (Part II., 1902).

NUMBER OF MACHINES, AND OF COLLIERIES WHERE USED, MOTIVE POWER EMPLOYED, AND QUANTITY OF COAL OBTAINED BY THEIR USE IN THE VARIOUS INSPECTION DISTRICTS DURING THE YEAR 1902.

District.	No. of collieries where machines are at work.	Number of machines.	Worked by		Coal obtained.
			Electricity.	Compressed air.	Tons.
East Scotland, ...	18	33	11	22	230,780
West Scotland, ...	19	52	7	45	526,033
Newcastle, ...	11	23	5	18	249,291
Durham, ...	12	30	20	10	223,109
York and Lincoln, ...	30	129	35	94	1,349,997
Manchester and Ireland, . .	14	23	9	14	80,036
Liverpool and North Wales, . .	18	70	4	66	418,161
Midland, ...	30	93	44	49	812,132
Stafford, ...	11	22	14	8	256,147
Cardiff	None	...	...	...	...
Swansea, ...	2	6	...	6	9,826
Southern, ...	1	2	...	2	5,690
Total, . .	166	483	149	334	4,161,202
1904	...	755	270	485	...

A large proportion—more than three-fourths—of the machines employed are of the well-known disc type, which seem to be best suited for British mines. The following table shows this :—

MECHANICAL COAL-CUTTERS EMPLOYED IN THE UNITED KINGDOM IN 1902.

Kind of Machine.	Number in use.											
	District No. 1.	District No. 2.	District No. 3.	District No. 4.	District No. 5.	District No. 6.	District No. 7.	District No. 8.	District No. 9.	District No. 10.	District No. 11.	District No. 12.
A, . . .	29	49	19	14	126	17	53	73	11	...	2	2
B, . . .	1	...	2	2	1	3	15	2	1	...	2	...
C, . . .	...	...	...	6	2	1	2	3	7	...	...	...
D, . . .	...	3	1	8	...	...	...	11	2	...	...	...
E, . . .	...	...	1	...	..	2	...	4	1	...	2	...
F, . . .	3	...	...	...	...	...	..	...	...	...	...	...
Total, .	33	52	23	30	129	23	70	93	22	...	6	2

A, Disc machines ; B, Pick (Ingersoll or Sullivan); C, Revolving bar; D, Rotary header; E, Toothed endless chain; F, not stated.

Classification of Machines.—Coal-cutting machines may be divided into the following classes:—

a. Heading machines, such as the Stanley heading machine.
b. Disc machines, such as the Gillott & Copley machines.
c. Chain machines, such as the Jeffrey machine.
d. Bar machines, such as the Hurd machines.
e. Percussive machines, such as the Ingersoll-Sergeant machines.

a. **Heading Machines.**—This class of machine has, up to the present, been used only to a very limited extent, the only machine employed with success in British mines being the Stanley heading machine.

It consists of an iron frame carried on two wheels set one in advance of the other. On this frame is fixed an engine with two cylinders placed vertically, the pistons being connected to the engine shaft by two cranks. This engine shaft carries geared wheels at each end, and through these is geared to the principal cutting shaft,

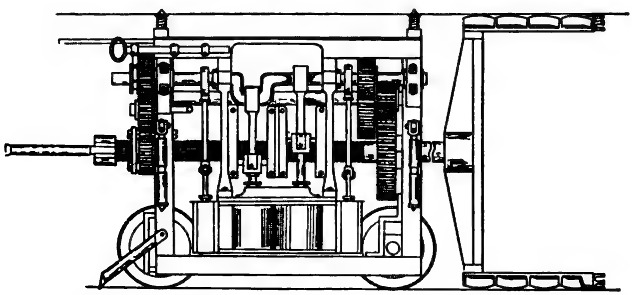


FIG. 99.—Stanley heading machine.

which passes through the centre of the frame. On the end of this shaft a cross-head is fastened, to which are attached, at right angles, two cutter bars upon which are fixed the cutters. These cutters revolve in a circle, and cut out a circular core of coal about 5 ft. in diameter. The main shaft has a screw thread cut nearly its whole length, by which, aided by suitable gearing, the cutters are advanced. The cutter bars project about 3 ft. beyond the cross-head; and this length controls the depth of each cut. The machine is 'anchored' to the sides and floor, by means of screwed arms, to maintain it in position, and to keep the cutters against the face. When a cut the length of the arms has been made, and the coal removed, the cutting motion is put out of gear, and the advancing motion put into gear, by which the whole machine is propelled forward ready to start a new cut. The whole machine, which is worked by compressed air, weighs between three and four tons. It can cut usually between 12 to 15 ft. in a shift of eight hours, including the time taken to bring down the coal and move the machine forward. Compared with hand labour,

the cost of cutting by this machine is high, but the great advantage of using it is the rapidity with which a field of coal can be opened out. It can only be used in thick seams, *i.e.* seams above 5 ft. thick.

b. **Disc Machines.**—There are now a considerable number of different machines of this type used in British mines, but they are all constructed on practically the same principle, viz., that the undercutting is done by means of a disc, on the periphery of which are fixed a number of picks or cutting tools, the disc working in the same way as a circular saw, but placed horizontally instead of vertically, and made to rotate by suitable gearing. These machines are exclusively used for longwall working, where they can be set to under-cut a long length of face in one operation.

The Gillott & Copley is one of the best known types of disc machine, and has been successfully used for over twenty years. It is worked by compressed air, and has a frame about 5½ ft. long by 2½ ft. wide, which carries two cylinders 8 or 9 in. diameter, and about the same length of stroke. The cutter-wheel, which varies

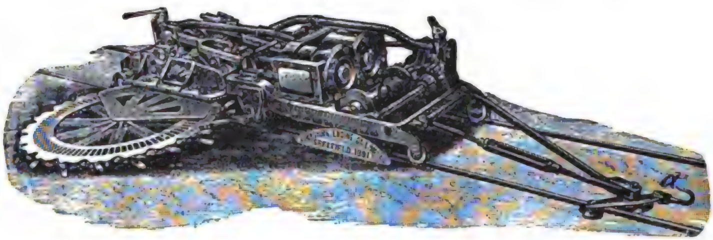


FIG. 100.—The Gillott & Copley coal cutter.

from 3 to 4 ft. in diameter, carries on its circumference about twenty-five steel teeth, and cuts to a depth of 3 to 3½ ft., the height of cut not being more than 3 in. This machine works at a low speed, the cutter-wheel making only about five or six revolutions per minute. The machine runs along the face on rails, and is fitted with the usual wire rope and drum arrangement. The disadvantage of this machine is that it does not cut level with the floor, the portion left on having to be taken up by hand. It can, however, be arranged to cut level, but it must be provided with a guard to keep the disc from taking the cuttings back into the holing.

Where the holing is done in the coal, this machine does very well, but is not so efficient when cutting in fireclay as some of the other disc machines. Where the under-cut only requires to be about 3 ft. or so, this machine gives good results under favourable conditions for working; where a deeper cut is required, some of the machines with larger discs, such as the Diamond machine, are generally used.

The Rigg & Meiklejohn machine is somewhat like the Gillott & Copley cutter, and holes the coal in the same way, the picks being

fixed into a circular disc. It is made so that it can cut at the very bottom of the coal. The construction of this machine will be understood by reference to fig. 101. It is placed on a steel frame *x y*, carried on four wheels, and kept in adjustment by screws at each corner. The two air cylinders *d d* are bolted to the frame, these cylinders being about 8 in. diameter and 10 in. stroke. A jib *b*, bolted to the side of the frame, carries the disc *a*, with eight or ten snugs *e e*, on which the picks are bolted. A bevel pinion *c*, with twenty-seven teeth on the engine shaft, is geared into a circular rack *f* of sixty teeth on the disc. The disc makes about sixty-two revolutions per minute, while the engine makes about 140 revolutions. The picks,

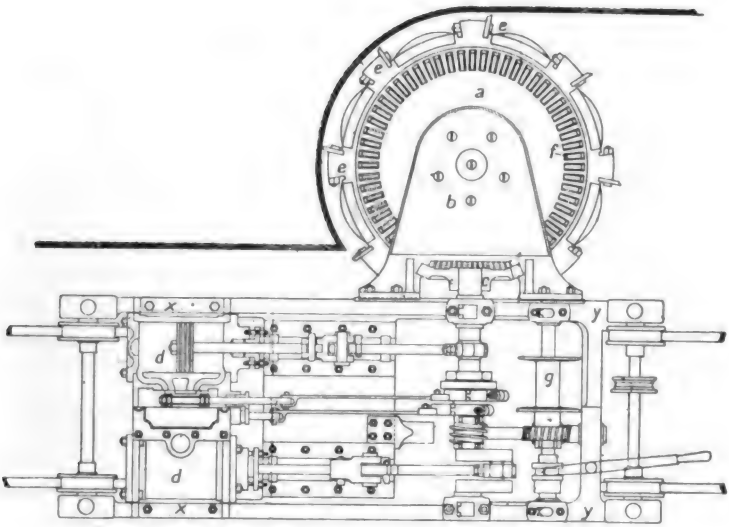


FIG. 101.—Rigg & Meiklejohn machine.

5 to 6 in. long, made of hard steel, are in sets of four, and shaped so that the whole thickness of holing, about 4 in., is fully occupied by the pick points. The disc revolves on a bearing, and is kept in position by a bottom plate, bolted with stud bolts to the jib. The machine is reversible, and cuts either way. The engines are reversed by turning the eccentric pulleys round on the shaft, no links being used. The machine, whilst cutting, is hauled along by gearing from the main driving shaft, on which there is bolted a split worm which works into a pinion on the under shaft. The under shaft extends from the engine sole-plate to the front of the machine in which the bearings are seated. A solid worm on the under shaft is geared into a pinion on the hauling shaft, on which the drum or chain wheel *g* is placed. A $\frac{3}{8}$-in. chain is coiled once or twice round the drum and

is made fast to a prop 60 ft. ahead. The depth of cut is about 3 ft. to 3 ft. 6 in., the height of the machine 22 in., and the weight about 18 cwt.

This is a very good machine, when compressed air is being used as the motive power, for thin seams, as in cutting it makes its own floor. It is equally suitable for cutting in either a coal or fireclay holing, and has been extensively used with successful results at a large number of collieries for a good many years. The motive power used is compressed air, but it is now being constructed to work also with electricity.

The Clarke & Steavenson machine (fig. 101A) is constructed on much the same principle as the last two described, with the exception

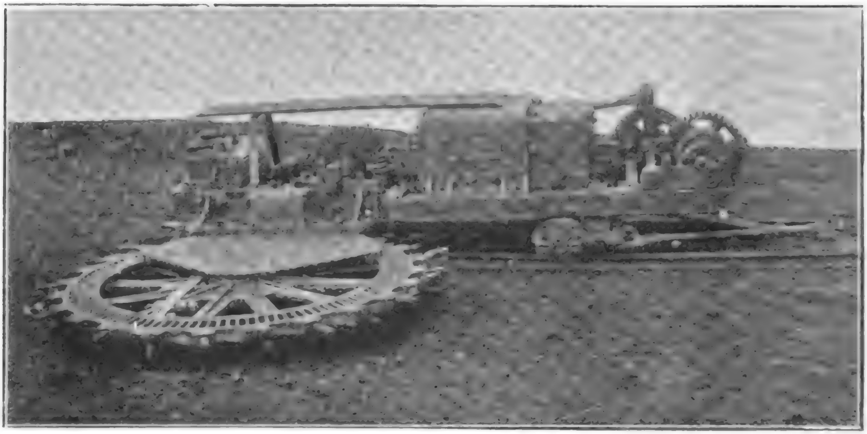


FIG. 101A.—Clarke & Steavenson machine.

of some minor details. Unlike the Gillott & Copley, or Rigg & Meiklejohn machines, it has been constructed to work with either compressed air or electricity as the motive power, the larger number now being made to be driven electrically. Two types of this machine are at present in use, the standard type for cutting in seams 26 in. and over, and the low type for cutting in seams under 26 in., the latter machines being capable of cutting in seams 20 in. in height. The Clarke & Steavenson machine, like those already described, is provided with a disc fitted with cutters which are alternately of the straight and forked type, or singles and doubles, and are secured to the disc by being fitted into boxes cast on its periphery and held in position by cotter bolts and nuts, or by split pins. The number of cutters or teeth in the cutting wheel is twenty, of which ten are straight, or singles, and ten forked, or doubles. The discs used are either 4 ft. or 6 ft in diameter, according to the depth of the under-

cut required, the former making an under-cut of 3 ft. 6 in., while the latter makes an under-cut 5 ft. 6 in. in depth.

The standard type is 2 ft. 2 in. in height, measuring from the top of the rail to the top of the casing, while in the special low type

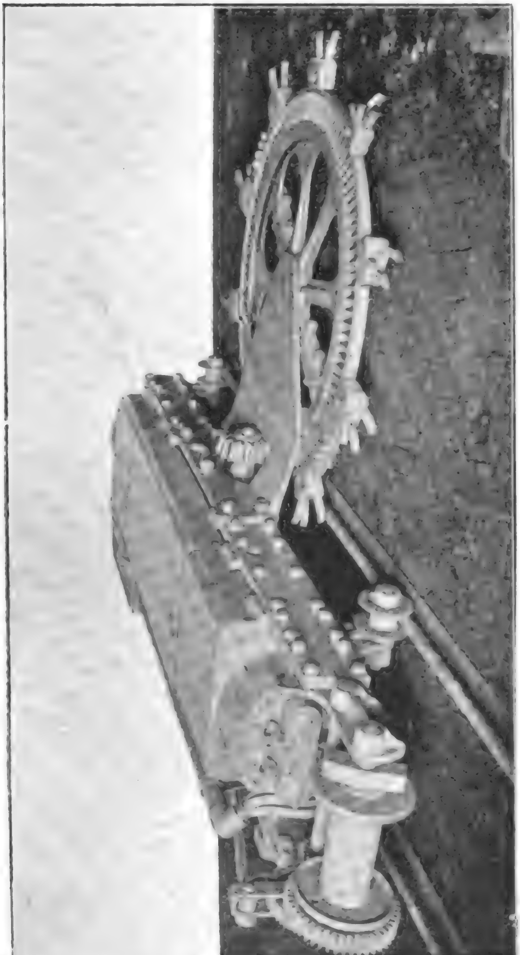


FIG. 101B.—Diamond machine.

the height is 1 ft. 5 in. In the Rigg & Meiklejohn machine the cutter wheel is placed almost at the front of the machine, but in the Clarke & Steavenson it is placed at the rear, which is a great advantage, as the material from the cut is delivered at a point close

to the driver. As a consequence, if the machine is working beyond its capacity and is hard pressed, the driver can materially assist it by clearing back the cuttings from the wheel. This at first was considered a disadvantage, as it was thought that it would prevent the machine from cutting forward and backwards if required, but it has been found that by changing the cutters in the disc (*i.e.* putting in cutters suitable for the direction in which the machine is to travel) the machine can cut back equally as well as forward. At the same time that the cutters are changed the long bridle must also be changed to the other end, and the haulage rope drawn through under the machine. The Clarke & Steavenson machine was first introduced about ten years ago, and since then it has proved to be a good all-round coal-cutter for working under very various conditions.

The Diamond machine (fig. 101B) is another well-known machine of the rotary disc type. A large number of these machines are at work in the Midland coal-fields, and have been found to give good results in seams where a deep under-cut is required. The Diamond Coal-Cutter Company, who manufacture the machine, were the first to introduce deep under-cutting, the depth of the under-cut varying from 4½ to 6½ ft., the latter depth of cut being employed in seams of medium thickness, *e.g.*, 4 to 5 ft. Diamond machines are also made to cut in thin seams of 2 ft. and under, with an under-cut of 3 to 3½ ft., as is usual in such seams. Two types of this machine are now constructed, those fitted with cylinders to work with compressed air, and those fitted with motors to work with electricity. Several new features have been introduced into these machines. In the compressed air machines the power-cylinders, instead of being placed side by side at one end of the machine, as is the usual practice in the other disc cutters described, are placed one at each end of the machine, an arrangement which has an important effect in assisting to balance the large cutter-wheel. This arrangement also decreases the width of the machine, which is a point of no small importance, especially where cutting with machinery has to be done in seams with a bad roof. Another feature of this machine is that by fitting on an extra pair of axles and wheels and turning the machine over, it can be made to cut at any height in the seam, and it can cut in either direction. The cutting wheel, which is of the usual disc pattern, is attached to a strong bracket fixed to the framework of the machine.

The Anderson-Boyes machine (fig. 101C) is another of the longwall machines of the disc type, which has recently been put on the market by the firm of Messrs Anderson, Boyes & Company, of Motherwell, N.B. The machine, which is electrically driven, has been mainly designed by Mr Daniel Burns, M.E., who has had a large practical experience in the installation of coal-cutting plant in Scotch collieries, especially thin seam collieries, and the design of the machine is the outcome of that experience, the improvements which he has intro-

duced being suggested by the defects in other coal-cutters which are now at work in our coal-fields. Of course, every machine has some imperfections, and the ideal machine to work under varying conditions is as yet in the experimental stage. The following is a description of the machine :—The frame, which is of heavy angle steel $4\frac{1}{2}$ in. by $4\frac{1}{2}$ in. by 1 in., extends along each side, and is braced by the motor box at one end and the switch box at the other, both being attached to the frame by heavy fitted bolts.

The cutter wheel is 4 ft. 6 in. in diameter, and carries twenty cutters, ten doubles and ten singles. The peripheral speed of the cutting wheel is about 300 ft. per minute, and the reduction gear is carried on three shafts. In order to keep the pinions closely in gear the plummer blocks are all cast in one piece, the arrangement of the gearing for the cutter wheel being such as admits of the use of a

FIG. 101C.—Anderson-Boyes machine.

short stiff bracket which carries the wheel and arc-plate. The haulage gear is fitted low down in the body of the machine just behind the switch box, and is arranged so as to take the rope upon the under side, whether the machine is cutting back or forward. An arrangement is fitted to the ratchet wheel which admits of the feed being thrown off or regulated while the machine is running. This is fixed in a position which is easily accessible to the person driving the machine, and is an appliance which should save a great deal of time where the cutting has to be done in under-clay that varies much in hardness. The most important part about the machine is the motor, which is specially constructed for the work, and is of much greater power than those most commonly used in coal-cutters, being 39 horsepower if the machine is series wound, the class of winding most generally used for coal-cutters; if shunt wound it gives 38 horsepower.

The Jeffrey machine is an American type of disc machine which has only recently been introduced into this country. It is constructed in two types, viz., to be driven by compressed air and to be operated by electricity. The construction of both types are nearly similar, in the latter a motor being substituted for the air cylinders. The electrical machine consists chiefly of two parts, the motor and feeding gear on its frame, and the cutting wheel. The frame of the machine is of the usual rectangular shape, and made of steel, with the haulage drum for moving it along the face placed at the forward end. The motor is placed in the middle of the frame, and the cutting wheel is placed at the extreme rear end. By means of a hand wheel the disc can be tilted up or down so as to follow any unevenness in the floor, or pass any obstruction in the holing, such as ironstone balls, which may be encountered. A device is also provided for altering the speed of travel of the machine without changing the speed of the cutter wheel. Another novelty in connection with this machine is that it only requires a single rail to run on, this rail being placed under the centre of the machine, the side thrust being taken up by special sleepers and light screw-jacks. The machine is operated from the front end, the feed being driven by an eccentric through a ratchet and pawl on to the haulage drum, and there is an arrangement for enabling the feed to be stopped, started, or adjusted without it being necessary to stop the motor. This enables the machine to clear itself should it get hampered by a fall of coal. Three rates of cutting are provided for—8, 16, and 25 inches per minute, or *vice versâ*, should such lowering of speed be required, according to whether the under-cut is soft or hard. The machine is constructed to cut either forwards or backwards, making it suitable for working in a short length of face. The high-speed wheels gearing the armature down to the cutting wheel and feed eccentric are enclosed in a casing arranged so as to run in oil, which reduces the noise of the machinery while cutting is being done, and so enabling the attendants to hear any movements in the coal or roof more readily. The cutters, which are all singles, are secured in position by suitable boxes on the periphery of the wheel. One half of the cutters are set with their faces upwards, and the other half downwards. Different sizes of cutting wheels are used according to the depth of under-cut required, the smallest size cutting 3 ft. 6 in., and the largest cutting 6 ft. under, the height of the cut being 4 to 6 in., according to depth. The electric machine is fitted with a 25 horse-power motor, and weighs about 34 cwt., the principal dimensions being 8 ft. 2 in. long, 3 ft. 8½ in. wide, and 19 in. high from the top of the rail.

Chain Machines.—These machines are of comparatively recent date, having been first introduced into American mines in 1894, but since that time they have rapidly made their way into a large number of mines in the United States. Only in a few instances have they been used in Great Britain.

They are most largely employed for cutting in narrow work, *i.e.* in pillar and stall workings, although machines like the Morgan-Gardner are suited for longwall work. The principle of the chain machine is almost that of the band-saw, with the addition of suitable mechanism for moving the machine while it is cutting, The machine consists of three principal parts:—(*a*) A fixed carriage which can be clamped fast to the side of the working place by screw-claws, the cut being effected laterally, the claw holding the carriage in front being set at an angle of 45° so as to take up the thrust; (*b*) a movable frame in the shape of a trapezium that rolls longitudinally on the carriage, the short base of which carries the motor, while the long base is in contact with the face, having over its whole periphery a groove through which passes the endless chain; (*c*) the endless chain, moving in the groove of the frame, passing at the back round a driving pinion, and in front over two small guide pulleys. The cutters, which are often set upwards and downwards alternately, are

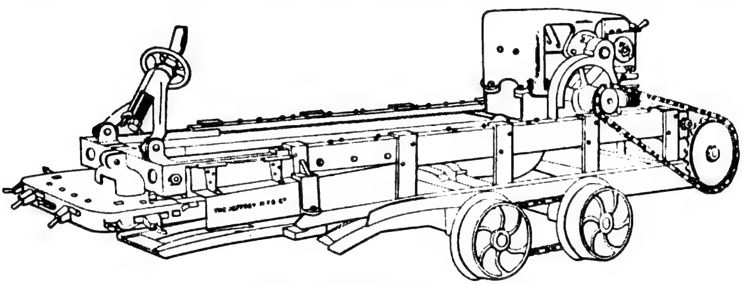


FIG. 102.—Jeffrey chain machine.

fitted to the links of this chain. Nearly all of these chain machines are now driven by electricity.

The Jeffrey chain machine (fig. 102) is designed to work either in pillar and stall or longwall workings, and can be built to work by compressed air or electricity. It consists of three principal parts, the bed frame, the sliding chain cutter frame, and the motor carriage. Upon the frame are mounted the feed racks and a cross-bar on which rests the jack for taking the backward thrust. The cutter frame consists of one steel centre rail, the cutter head, and two side guides for the cutter chain. The cutter frame is triangular in shape, making it necessary to use only three wheels, two in the cutter head and the sprocket wheel, for conveying power to the cutter chain. The driving and feeding mechanism, consisting of steel pinions and wheels for working the machine, are mounted on the carriage. The cutter bits, which are fixed in the ends of the cast steel cutter links, are straight, with a slight hook at the cutting end. When it is set working, the chain begins to travel round, while at the same time the cutter head and frame are advanced into the coal. The machine

having cut to the full depth, the direction of travel is reversed and the cutter frame drawn in again. The machine is then slid along a distance about equal to that of the cutter head, and the work proceeds as before. The machines are built to under-cut $5\frac{1}{2}$ and 7 ft., the height of cut being 4 to 5 in.

Two types of the Jeffrey machine are constructed, the ordinary type for working in moderately thick seams, and the low type constructed for operating in thin seams. The construction of both machines is practically alike, the only difference being that in the low type the motor is set much lower down in the carrying frame, which reduces its height.

Another type of the chain machine is the Morgan-Gardner, which in construction resembles very much that of the Jeffrey machine, with the exception of some minor details, and so does not require further description.

Another machine of the chain type which has been recently introduced is that known as the Mather & Platt machine, manufactured by the well-known firm of that name at Manchester. The construction of this machine is much on the same lines as those just described.

Chain Shearing Machine.—This type of machine is used for shearing or side-cutting the coal in narrow places, such as in pillar and stall working. The construction of the machine is somewhat like the chain under-cutting machine, but with the frame round which the cutter chain passes placed in a vertical instead of a horizontal position. The outside frame is constructed of heavy steel angles with the flanges turned in, which gives a broad wearing surface for the carriage that carries the motor and gearing. The machine is fitted with a motor at the rear end of the machine which drives the shearing chain through spur and bevel gearing.

Bar Machines.—This is a type of machine which is very different—so far as the tool used for under-cutting is concerned—from the two classes of machines just described. Instead of the under-cutting being done by a disc or moving chain, a circular tapered steel bar with a number of teeth or cutters fixed in its periphery is employed to do the cutting. For under-cutting in certain seams this type of machine has some advantages over the disc cutter. In seams which are soft and friable, if disc machines are employed a good deal of trouble is often experienced by the under-cut coal coming down on the disc before it can clear itself, or before sprags or holing props can be put in to support the cut coal. The larger the disc used, the more likely is there to be trouble from this cause, as the area of under-cut and unsupported coal will vary as the diameter of the cutting wheel. These falls of coal taking place on the disc cause delay, as the machine requires to be stopped until the fallen coal is cleared away; and where cutting is being done in thin seams with a bad roof where propping is carried close to the machine, the clearing of such coal is often a tedious and difficult process. Where such conditions prevail, the bar machine may

be employed with good results, as only a small area is under-cut and left unsupported for a very short time, as the spragging can be done quite close to the cutter bar. Another advantage that is claimed for this type of machine is that less power is required to drive it than a disc or chain machine for an equal depth of under-cut. In actual working practice we do not think there is a very great difference in the power required for driving the two classes of machines, and it is a point on which at present too much value need not be put, for the power required for driving is after all not of so much importance as the time required to cut a given area by a machine.

Other advantages claimed for the bar machine are that there are no bearings under the coal when cutting is proceeding, as in the centre of a cutting disc; that the machinemen are not so hard worked as

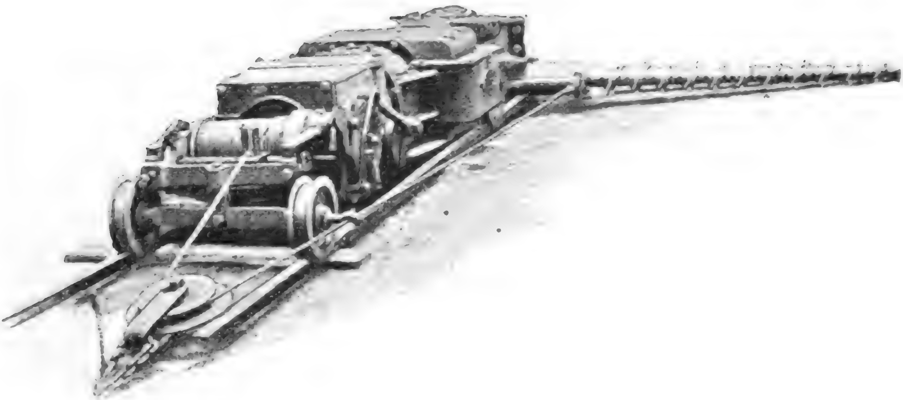


FIG. 102A.—Hurd machine (under type).

with disc machines; that the holings are better cleared out; and that the cutters are easier examined.

The Hurd bar machine (figs. 102A, 102B) is the best known machine of the bar type. It has been installed at a considerable number of collieries in England and Scotland, but its numbers are limited compared with the number of disc machines which are now in operation.

The Goolden bar machine is practically similar to the Hurd in its general arrangements, with some differences in detail. In the Hurd machine the cutter-bar has a reciprocating as well as a circular motion, which produces a chipping as well as a cutting action, while in the Goolden machine the bar has only the circular motion.

Percussive Machines.—The percussive machine, as its name implies, under-cuts the coal by a series of blows in much the same way as a

miner would under-cut with his pick, so that their action is altogether different from the machines previously described in which the under-cutting is done by a continuously rotating disc, chain, or bar. Percussive machines are, therefore, practically pneumatic or electric picks. They resemble mechanical rock-drills in that they are only furnished with a single cutting bit, though they differ from them in having no mechanism for rotating nor a screw for giving the feed, while, instead of being fixed at work, they are essentially movable. These machines were first introduced into American mines about 1880, and since then they have been extensively employed for mechanical cutting in the coal mines of the United States and Canada. In quite a large number of the mines in those countries the under-cutting is entirely performed by percussive cutters, although in some districts they are being gradually displaced by the chain machine. Up till the present they have been very little used in British mines, the other

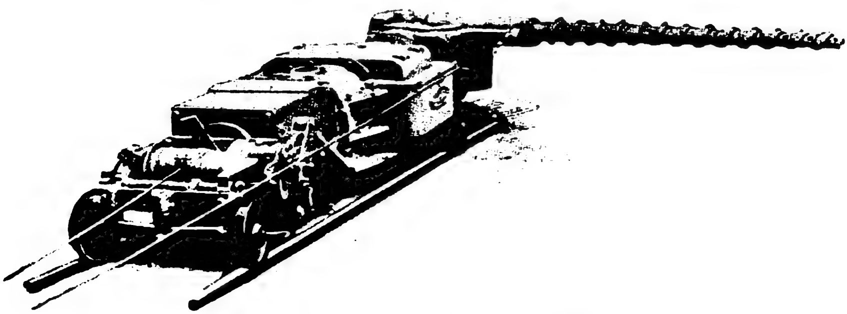


FIG. 102B.—Hurd machine (over type).

types of machines being preferred. The American coal-seams are more suitable for cutting by percussive machines, being of a softer and more friable nature than the coal-seams of Great Britain. All percussive machines are of one type and are worked on a similar principle, though differing somewhat in detail from each other.

They are constructed with the following main parts * :—

(*a*) A cylinder 3 to 4 in. diameter and 9 to 12 in. stroke.
(*b*) A piston and rod, with the rod extended to make the cutting tool.
(*c*) A valve and valve-chest, the valve being thrown by air, or operated by a small independent engine, cushioning arrangements being in all cases a necessity.
(*d*) A sleeve or guide attached to the front end of the cylinder, to guide and steady the piston-rod, and also provided with arrangements for preventing the piston and cutting tool from turning.
(*e*) A suitable pick or cutter, usually sharpened to a fish-tail shape.
(*f*) A pair of plain wheels upon which the machine is carried, the wheels being placed so as to balance the machine.
(*g*) A pair of handles at the back for the machineman to take hold of.

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* *Engineering Magazine*, August 1903, p. 125.

Method of Working Percussive Machines.—"In operating a pick machine the runner sits on a board or platform, inclined to the face of the coal; one foot of the operator is braced against one wheel of the machine, and with the two handles he directs it against the coal, picking off the coal exactly as a miner would do, except with much more force to each blow. The under-cut made is V-shaped, 12 to 18 in. in height at the face, and tapering back to a feather edge on the floor at the rear of the cut, the depth of the cut being from 3 to 6 ft. deep, according to the thickness of the coal. A helper shovels away the cuttings as the machine, guided by its operator, loosens the coal in the under-cut." *

The work in operating these machines is severe, especially with unskilled machinemen and in hard coal, for the inclined board and the chock provided by the operator's foot only partially takes up the recoil of the machine. In a hard seam, unless the machineman can rest frequently, the labour of keeping the machine up to its work becomes very fatiguing, in fact almost unendurable, and hence, as already stated, they are best suited for cutting in soft and friable seams.

There are now quite a number of these machines in use, but we need only describe one or two, as they are all so much alike.

The Ingersoll-Sergeant machine (as also the Harrison machine) is fitted for working in narrow "rooms" or drifts. It is simply a chisel attached to the continuation of the piston-rod of an air cylinder, and may be called a slotting machine. The machine (fig. 103) is mounted on a pair of wheels to enable the cutting tool to operate in any direction, either under-cutting or cutting in an upward direction, and to facilitate the removal of the machine from point to point. The tool receives a reciprocating motion from a small cylinder of the usual type fixed to the frame of the machine. The piston works through suitable packing at the front end of the cylinder, and is made long enough to project beyond the end of the sleeve in which it works when at the end of its back stroke. At the forward end of the cylinder is placed a buffer of leather of such construction as to form a cushion for the piston. The projecting part of the piston-rod is tapered so as to fit a corresponding taper in an extension piece which is fixed to the piston-rod by a key. The outer end of this extension piece is made to receive the pick or chisel, which is also held in position by another key. The piston-rod is provided with eight straight grooves engaging in similar projections in the end bushing for the purpose of preventing any accidental movement of the piston and pick. The cutting edge of the pick is shaped like a V, having a sharp edge and cutting points. The pick can be easily removed and sharpened. The machine, as already described, is worked from an inclined board of convenient size, while making an open channel under the coal of 4 ft. to 5 ft. of face and 3 ft. to 5 ft. of under-cut. The claims of

* *Mines and Minerals*, June 1903, p. 510.

this machine are that it is small, simple in construction, requires little space, is light—weighing only about 750 lbs.—portable and cheap.

The Harrison machine is very similar to the Ingersoll-Sergeant, but is somewhat heavier and stronger. The air distribution is effected by a small auxiliary motor, which cuts off a half-stroke, and the

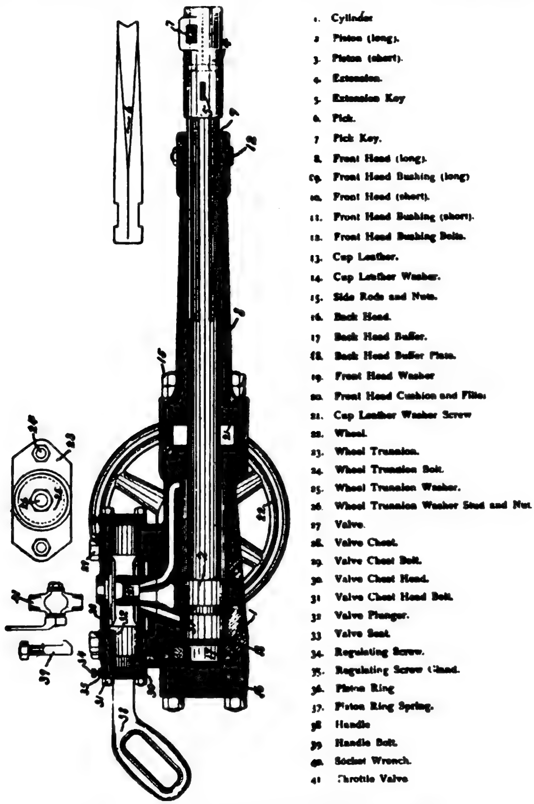


FIG. 103.—Ingersoll-Sergeant machine.

cylinder head is protected by a leather cushion, which deadens the blow of the piston when the bit or cutter does not strike the coal. With an air pressure of 70 lbs. per square inch this machine gives 190 to 210 blows per minute, with a stroke of 11 in. It makes an under-cut of 6 ft. deep and about 6 in. in height.

The Sullivan machine, which is also driven by compressed air, has a system of variable expansion that permits of graduating the blow, and the cylinder head is protected by an air-cushion. It greatly

resembles an ordinary rock drill, is mounted on a carriage running on rails, and the bit, much longer than usual, is supported by an arm that also acts as a guide. This appliance can be moved in a vertical plane so as to make a cut in the face for its whole height; and the forward feed is given by a small winch on which a chain is wound, so that a cut 6 in. wide can be made to a depth of 7½ ft.

The Morgan-Gardner percussive machine, unlike the three above described, is driven by electricity, and is practically the first of its class which has been successfully worked by electrical power. The construction of the machine is practically similar to those just described, with the exception that an electric motor is substituted for a compressed air cylinder. The rod with the cutter attached is drawn backwards by a cam during half a revolution, and then a strong spring propels it sharply forward, the cam being worked by gear from a continuous current motor with vertical axis carried on the machine. It can give from 175 to 225 blows per minute, and cuts to a depth of 4½ ft. The weight of the machine is about 750 lbs., and it has a total length over all of 7 ft.

The Champion machine is also of the percussive type, but differs very considerably in construction from those just described. It is practically a combination of the percussive machine and rock drill. The machine consists of five essential parts, viz.: (1), The supporting column (5 ft. long), 200 lbs.; (2), the segment with accessories, 114 lbs.; (3) the air drill, 239 lbs.; (4), extension rods, two, for cutting up to 7 ft. deep, 28 lbs.; (5) the bit. The supporting column, of simple construction, is securely fixed about 3 ft. from the working face. Sliding on this column longitudinally is a sleeve which can be fastened at any given height of the support. In a bearing of the sleeve the toothed segment is carried, and may be rocked therein from a horizontal to a vertical plane, or to any intermediate position. Pivoting in the hub of the segment and in its axis is a connecting piece commanded by a worm acting on the teeth of the segment; a handle driving the worm may be attached to either end of its axis. The drill machine is securely fastened to this connecting piece, and can therefore be swung in a plane parallel to the segment by turning the handle of the worm; if, therefore, the segment is attached horizontally, the machine will act as a coal cutter; if vertically, as a coal shearer; if at an angle, the machine will cut into the coal at that angle. It will thus be seen that the machine can make a cut at any given height and at any given angle. It can also be used as an ordinary rock drill, and this simply by not turning the handle commanding the worm. The working of the machine is controlled by one man, who with one hand turning the handle of the worm produces a swinging action of the drill, while with the other hand he regulates the advance of the drill into the coal by turning, as in any ordinary rock drill. When the

drill is set to work, the cutting-bit strikes the coal at the rate of about 350 blows per minute. It would penetrate very fast into the coal were the blows directed to one place only; but, while the machine is at work, the cutting-bit is gradually displaced in a plane parallel to the segment, by turning the handle of the worm; and the cutting-bit therefore strikes every blow in close proximity to the preceding one. It describes an arc having for its centre the axis of the segment, and the outcome of this motion is an even cut. When the cutting-bit has reached the end of an arc, the drill is advanced to the extent it has been cutting into the coal by turning the feed screw, and the motion given to the worm is then reversed.

Percussive machines are best suited for thick seams with a good roof, as they require a clear space of 7 to 9 ft. In the American mines their use is almost exclusively applied to narrow work in pillar and stall workings, but they could also be usefully applied in conjunction with longwall machines in cutting the narrow work which is necessary to allow most disc machines to start work, this narrow work having at present to be done by hand labour.

Choice of Machine.—If the owners or managers of a mine determine upon adopting mechanical cutting, the first question which will naturally arise is, what type and class of coal-cutting machine should be used for the work to be done? This is not so simple a question as it looks, and cannot be answered right off-hand. The choice of machine will depend upon various circumstances, such as—(1) The mode of working at the mine; (2) the nature of the holing; (3) the position of the holing, *i.e.* whether the holing is to be done at the bottom, centre, or top of the seam; (4) the thickness of the seam; (5) the nature of the floor and roof.

If the mode of working is longwall, which is most usual in this country where coal-cutting is done by machinery, then the selection of a machine must be made from the disc, bar, or chain classes of coal-cutters. We have already stated that by far the larger number of machines used in British mines are of the disc type, and under ordinary conditions this seems to be the type most suitable, but, as already pointed out, there are certain kinds of seams in which it would be possibly better to adopt the bar machine, *e.g.*, where the coal is soft and friable, or where the holing has to be done in the centre or at the top of the seam. Hitherto chain machines have been little used in this country, and on the whole they do not seem to be so well suited for the seams to be cut here as in the American mines where more favourable conditions prevail for mechanical cutting. As a rule chain machines take up too much space for our thin seams, and are too clumsy to handle. Before making choice of a particular machine, the whole circumstances under which it is to work must be carefully studied, and a suitable type of machine introduced to meet the particular requirements at the colliery, for a machine which

gives excellent results in one seam might prove a failure in another. The question of the best machine for any given seam is often only solved by actual trial and experience, as different seams may require different types of machines to get the best results under the prevailing conditions. Previous to selecting a given type or class of machine, it would be well for the interested parties to visit a number of collieries where different kinds of coal-cutters are in use, and to compare carefully the conditions under which they are working with the conditions in the seam or seams which it is proposed to cut by machinery.

Conditions suitable for Coal-cutting Machines.—As we have pointed out, the question of whether a certain seam is suitable for being cut by machinery cannot be answered off-hand. Some seams are more suitable for cutting by machines than others, while others may not be suitable at all to work with coal-cutters, for it must be distinctly understood, as things at present stand in Great Britain, all seams are not suitable for mechanical coal-cutting. Under certain circumstances coal can be got in better condition and at a cheaper rate by machinery than by hand labour. These circumstances may be stated thus:—

1. When the coal or underclay is of a hard nature and not easily holed by hand labour.
2. When the seam is under 3 ft. 6 in. in thickness.
3. When there is a good roof and floor, the latter being fairly level.
4. When the wages of miners are high.

While we have stated that seams under 3 ft. 6 in. may be worked more profitably by machines than by hand labour, we would not say that seams of greater thickness cannot be worked to advantage with coal-cutters, for as a matter of fact seams thicker than 3½ ft. are being now worked at less cost with machines than by hand labour. What we do state is that coal-cutting machines can be used more effectively and more profitably in seams under 3½ ft. thick than in those of greater thickness. Regarding the condition of the roof and floor, there can be no doubt that machines require that both should be fairly good, especially the roof, for with a bad roof a considerable amount of extra labour and expense is involved. While, however, a good roof is a great advantage, it does not follow that with a bad roof it is impossible to cut by machines, for if due care is taken in propping and strapping, the difficulties of a bad roof may, in a large measure, be overcome. The floor should be fairly even, for it is not an easy matter to get the machine to follow the inequalities which are present in some seams. Seams that are much cut up by hitches, faults, and wants are not, as a rule, suitable for coal-cutting machines. Then as to the inclination of the seam, there can be no doubt that the best results are attained in seams that are level or with a low inclination, but if the workings are properly laid out for

mechanical cutting, the gradient is not a great difficulty; indeed, machines will be found at work cutting in seams with gradients up to 1 in 4 or 1 in 3. In cutting down such steep gradients the machine will require to be controlled by a back balance or a brake applied to the wheels of the machine.

Timbering at the Machine Face.—Timbering in a machine face is, in most cases, simple enough, especially when the roof is fairly good. The props are usually set up in rows, parallel to the roadway, at regular intervals along the face, a space of 3½ or 4 ft. being left between the front row and the coal to enable the machine to pass along. If the roof is of such a nature that it would be dangerous to leave such a width unsupported, straps or planks are wedged or notched into the coal-face at one end, and the other end supported by a prop. When the machine passes forward, other props are set up to the end of the strap supported by the coal. Sometimes light steel bars are used in preference to the wooden straps.

When the roof is bad and the coal cannot be relied on to give the necessary support to the end of the cross-straps resting upon it, a longitudinal bearer set underneath and at right angles to the cross straps, and parallel to the face, will require to be used. This longitudinal bearer may be of planking, or it may be a light steel channel girder. This method has been used at some collieries and found to answer the purpose well. The girders are used in 15 ft. lengths and weigh 25 lbs. per yard.

Props have to be set at regular intervals to these longitudinal bearers before the coal is cut, and as the machine advances they are removed one by one and re-set behind it. By adopting this system mechanical cutting can be carried on under roofs of a very bad nature. Chocks are also used in some workings, in addition to the props, for supporting the roof.

Labour for Machine Cutting.—To work a coal-cutter, usually three men are employed: one, a principal man, to drive the machine, shovel the small coal out from the cutting wheel, sprag the coal, and lift the rails behind and pass them up to front of the machine. This man ought to have sufficient mechanical knowledge to enable him to do any small repairs in the event of a stoppage or breakdown, and to get a man of this description, sufficient inducement, in the shape of good wages, will require to be offered. Other two men are employed in front of the machine, laying the road, putting in the necessary struts, attending to the bridle and hauling rope, and looking after any other work required during cutting. If the roof is bad, sometimes a fourth man is set specially apart to look after the timbering of the face.

Apart from the machine, another set of men are employed during the daytime. These are distributed as follows:—(*a*) A set of men for taking out the sprags, bringing down the coal, and backing it out, these being generally called hewers; (*b*) a set of men to get

the coal out to the road head (*i.e.* where the tubs are not taken along the face) and filling it into the tubs; (*c*) a set of drawers who draw the tubs out to horse or mechanical haulage. Where the tubs are lifted from the face by horse haulage this last set of men will, of course, not be required. It will also be found advantageous to employ a specially appointed man to supervise and be responsible for the whole of the work. Regarding the payment of all these sets of men, machinemen included, it will be found, as a rule, most satisfactory to pay them on the piecework system, *i.e.* pay them a tonnage rate for the cutting and getting of the coal. In some collieries the whole of the work is done by contract, but in most of the districts the miners' Unions are averse to this system, and give much trouble when it is attempted. It need hardly be said that the best class of men available should be employed for machine cutting, and especially for doing the actual cutting.

Motive Power for Machines.—The motive power employed for driving machines may be either compressed air or electricity. Up till the present time by far the larger number of machines have been driven by compressed air; but where new installations of coal-cutting plant are being laid down, electricity is being largely employed as the motive power. Each of the systems have their advantages and disadvantages.

Mr Garforth * states that "after twenty-eight years' experience with compressed air motors in underground work, it is the only power that can be safely used in certain deep gaseous mines, where the natural difficulties are quite sufficient without the introduction of artificial dangers. On the other hand, there are many mines which can be worked safely, and with advantage, by electricity." There can be no doubt that many mining engineers and colliery managers are averse to the use of electrical plant in fiery mines, but we think the dangers connected with the employment of electricity under such circumstances have been greatly exaggerated. Even in what are termed fiery mines it is seldom that an explosive atmosphere exists at the coal face where the machines are at work, and where sparking would be likely to take place. With the newer types of enclosed and gas-tight motors sparking is reduced to a minimum, and seldom occurs. It has, however, still to be proved whether electricity is more economical than compressed air as a motive power for coal-cutting machinery, but wherever electricity has been used the consensus of opinion is, so far, in its favour. A very large number of our mines are non-fiery, and in these there is no reason why it should not be used more extensively. With electrical coal-cutters, however, it requires more highly-trained men to operate the machines than when they are driven by compressed air.

One great objection to air-driven machines is the amount of noise made by the exhaust and the clouds of dust which it raises in dry and

* *Trans. Inst. Min. Engs.*, vol. xxiii. p. 338.

dusty mines. This is due to the fact that with some of the machines, such as the Gillott, and others, the exhaust blows directly on the floor. The noise and dust can be overcome to a certain extent in such cases by fixing a baffle plate under the exhaust. Compressed air has the advantage of being used with perfect safety, and any disarrangement can be easily repaired. On the other hand, the higher efficiency, lower cost for extensions, and greater ease with which extensions can be made with electricity, render it particularly suitable for coal-cutting machinery, and there can be little doubt but that, in the near future, it will be more extensively employed for such work, and will gradually displace compressed air as a motive power.

Cost of Coal-cutting Installations.—The cost of coal-cutting machines varies according to the type and motive power used. Disc machines driven by compressed air cost from £250 to £300 each; machines driven by electricity cost from £300 to £450 each. The cost of the generating plant on the surface will vary according to the number of machines which it is intended to operate; the cost of a generator for a single machine will be greater in proportion than if a number of machines are at work. When more than three machines are at work the cost of the generating plant may be set down approximately at £1000 per machine if compressed air is the motive power used; electrical plant will average somewhat more than this, probably £1100 to £1200 per machine. In these estimates the cost of piping or cables is not taken into account.

CHAPTER VII.

TRANSMISSION OF POWER.

Location of Machinery.—The power employed for driving machinery underground is very rarely generated in the workings, except in the case of steam, when boilers are sometimes placed underground, a method which ought not to be employed.

The power plant is, as a rule, placed at the surface, and the power transmitted underground: (1) By rods of wood or iron; (2) by wire ropes; (3) by steam; (4) by compressed air; (5) by hydraulic or water pressure; (6) by electricity.

Rods of wood or iron are chiefly employed in the case of pumping machinery, and ropes in the case of hauling machinery and dook pumps placed a considerable distance from the pit bottom. As a general rule it is not economical to work pumps by wire ropes, especially when the power is taken off the haulage ropes, unless the pump to be so actuated can raise all the water accumulated during the time that the hauling ropes are at work. It would be a great waste of power, and cause much wear and tear, to keep a long haulage rope in operation merely to work, perhaps, a single pump. There are, however, cases where pumps can be worked economically by wire ropes, especially if they are connected with the pump-rods in the shaft, or are otherwise apart from the haulage arrangements.

The transmission of power by rods is described in the chapter on pumping, and need not be referred to here, while the transmission of power by ropes is fully described under the heading Haulage.

Steam is used direct at a great many collieries for underground work, such as for driving pumps, haulage engines, etc., and if the distance is not too great, it is probably one of the cheapest and easiest methods of transmitting power. The maximum distance to which steam can be carried underground and applied with economy would seem to be about one mile, and the total loss of power at this distance may not exceed 20 to 25 per cent. if the pipes are carefully covered by a good non-conducting material, and the plant operated continuously so as to keep the pipes full of steam.

At Niddrie Colliery, near Edinburgh, steam is carried down a steep gradient, the inclination varying from 55° to 75°, for a distance of 3000 ft. to work winding and hauling engines and pumps placed at various parts of the workings. With a steam pressure at the

surface of 45 lbs. per sq. in., the gauges in the mine recorded from 41 to 42 lbs. per sq. in., or a loss of only 3 to 4 lbs., which is an exceedingly small percentage of loss when the distance the steam is carried is taken into account. But while the loss in pressure was small, the loss in other ways was shown to be fairly large, owing, no doubt, to the intermittent working of the haulage and winding engines. There are twelve boilers, one or two of which are, however, always off for repairs, cleaning, etc. When the whole plant is in operation, 3400 gallons of water are evaporated per hour. When all the machinery is idle, but steam is on, and cylinders, steam pipes and relief-valves open, it requires 1028 gallons of water per hour to maintain the pressure and keep the water in the boilers up to a fixed point. This shows a dead loss of 30 per cent., most of which takes place in the underground piping and machinery.*

The greatest source of loss in conveying steam to underground workings is due to condensation, and to minimise this, high-pressure steam should be used in supply pipes of small bore. At Bockwa Colliery, Germany, the pumping engine is placed at a depth of 590 feet from the surface, and is designed to raise 3300 gallons per minute. The steam pressure generated at the boilers on the surface is 150 lbs. per sq. in., the steam being conveyed to the engines in two columns of wrought-iron pipes each 4 in. diameter. These pipes are jacketed with a preparation composed of cork, bound round with zinc wire, and coated with gypsum, jute, and asphalt, and an outside sheathing of canvas, painted with Stockholm tar. In this way the loss by condensation is reduced to a minimum.

The steam pipes should be carefully fixed in the shaft and properly guided, the pipes being led between two rollers, which are more suitable than rigid guides. Some arrangement should also be made to cut off steam by an automatic runaway valve, in the event of a pipe bursting, as serious danger would result from the escaping of a large volume of steam into the workings before the supply could be shut off from the boilers at the surface. Great care should also be taken in making the connections, as leaking joints are a source of much loss of power and often of danger. Joints made with asbestos rings are preferable to those made with india-rubber.

The disadvantages attending the employment of steam underground are—

Loss of pressure due to condensation and leaking joints.
The danger of the sudden bursting of a pipe in the workings, or the blowing out of a joint.
The discomfort from the heat due to the great increase in temperature of the air in the mine, particularly in narrow, confined roads.
The bad effects of the moisture, due to the steam, on the roof and timber.
The difficulty of dealing with the exhaust steam, particularly if the engine is at a long distance from the shaft.
The danger of fire when pipes are led in confined places.

* *Trans. I. M. E.*, vol. xv. p. 264.

Compressed Air.—As a medium for the transmission of power to machinery in underground workings, compressed air is very suitable. The ease with which it can be conveyed to distant parts of the workings, the absence of heat in the pipes, and the beneficial effect it exerts on the ventilation, make it invaluable for certain classes of work, such as driving coal-cutting machines, rock drills, hauling engines, and pumps; indeed, for rock drills no other motive power presents anything like the same advantages. Owing, probably, to the large first cost for compressing machinery, and the low efficiency obtainable—25 to 40 per cent.—it has never been used to the extent it might otherwise have been had the efficiency been higher, because once the plant is put down, the working expenses are not excessive compared with an ordinary steam engine.

The cost of a compressed air plant, with coupled horizontal steam cylinders 22 in. diameter and 3 ft. stroke, and air cylinders 24 in. diameter and 3 ft. stroke, with steam boilers, air-receiver, main pipes, etc., complete, should not be more than £3000 or £3500, which would include its erection and preparing foundations.

Air compressors are simply force pumps in which the air is drawn into the air cylinder by an inlet valve, and is compressed and forced through an outlet valve into an air receiver, from which the supply is drawn for the underground workings. The usual arrangement is to have a pair of steam cylinders placed horizontally and coupled direct, with a fly-wheel in the centre. The piston-rods are continued through the cylinder, and connected to two air cylinders also placed horizontally and in a direct line with the steam cylinders.

Compressors are usually of three classes: (1) Dry compressors; (2) wet compressors; (3) injection or spray compressors.

Dry Compressors.—These are extensively used for colliery work, and give fairly good results if fitted with a water jacket and if the air pressure is not too high. A dry air compressor in its simplest form consists of an ordinary cylinder provided with a tight-fitting piston and suitable valves for admitting and delivering the air.

During the out-stroke of the piston, air rushes in and fills the cylinder through the valves; as soon as the piston commences the return stroke, the valves close and the air in the cylinder is compressed until it lifts the delivery valves, when it is forced out of the cylinder into the receiver. It will be seen from this that the action is exactly the same as in an ordinary pump. In Fowler's dry compressor air enters through the inlet valves as the piston moves forward; on the return stroke the air is delivered through the outlet valves, the cylinder being water-jacketed to keep it cool.

Another dry compressor of a quite different design is the Ingersoll-Sergeant compressor. The air does not enter the cylinder in the ordinary way, but first passes through a hollow tail-rod, the inlet valves being placed inside the piston. The outlet valves are placed

at the ends of the cylinder, while a water jacket surrounds the cylinder, being kept cool by a continuous flow of cold water.

The advantages claimed for this compressor are :—

The air may be taken from whatever point is most favourable by dryness, reduced temperature, and freedom from dust.

The admission of air being through a single tube, creates a constant and uniform draught in one direction only, thus filling the cylinder at each stroke with air at atmospheric pressure.

The construction of the valves admits of a large area of inlet with but a small throw of valve, allowing the compressor to be run at a high speed.

There being no inlet valves in the ends of the air cylinders, the space otherwise occupied by these valves is filled with cold water, thus presenting a cooling surface to the compressed air near the end of the stroke when the air is hottest.

Wet Compressors.—Wet compressors are used to a considerable extent, but in this type a large volume of water has to be put in motion at each stroke. This absorbs a large amount of power without any recompense, and the engines must also move at a very slow speed, hence large engines involving increased expenditure must be used. Moreover, the column of water in the cylinder, by repeatedly moving backwards and forwards, soon gets hot, and loses the advantages which it is meant to confer, namely, to cool the air in the cylinder during compression. Where large engines are available, and only require to move at a slow speed—not more than 40 or 50 ft. per minute—wet compressors may be used sometimes with advantage.

Injection or Spray Compressors.—Instead of using a solid column of water inside the cylinder to absorb the heat from the air, a supply of water in the form of fine spray is injected into the cylinder. This cools the air, which is now carried forward into the receiver. In the Dubois-François compressor (fig. 104) the spray and wet compressor are combined. It consists of an ordinary pump having large chambers *a a* at each end of the cylinder. The air is admitted through the inlet valves *c c*, and delivered through the outlet valves *e e*. A fine spray of water is injected into the cylinder through the small pipe *d*, and coming in contact with the air, cools it. This type of compressor, like the ordinary wet machine, requires to be run at a low speed, less than 150 ft. per minute, which means that the compressor must be large for a given output of air. Dry compressors are usually run at a speed of 350 to 500 ft. per minute, hence the preference shown for this type of machine. The introduction of water into the cylinder in any form is a very defective method of cooling the air, and often does more harm than good, as it may corrode the walls of the cylinder.

The best method of cooling the air is that of *stage compression* with intermediate cooling. Compressors made on this principle are usually either single- or double-stage compressors; double-stage compressors are, however, only used where great economy is required,

and where the power generated is above 150 horse-power. In this system the air is first subjected to low pressure in one cylinder, passed through an intermediate cooler, and thence to a second cylinder, where it is then compressed to a higher degree. The initial cost of either double-stage or triple-stage compressors is very

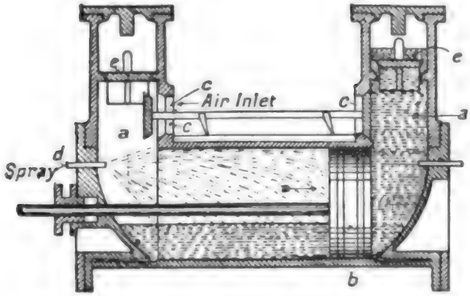


FIG. 104.—Spray compressor.

much greater than for ordinary compression machinery, and stage compression is only suited for the production of pressures above 60 lbs. per sq. in. In such circumstances a great saving may be effected, as 20 to 25 per cent. higher efficiency will be obtained than by ordinary methods of compressing air. In an installation at Paris, Professor Riedler combined stage compressing with spray injectors, which resulted in saving two-thirds to three-fourths of the work

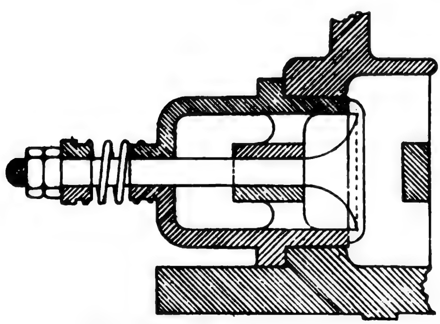


FIG. 105.—Mushroom valve.

expended in uselessly heating the air, the loss due to heating only amounting to 12 per cent. The Riedler two-stage compressor in this case gave a useful effect of 77 per cent., allowing 0·85 as the mechanical efficiency. With stage compression the engines can be worked at a much higher speed, as they are better balanced.

Valves for Compressors.—A great many different types of valves are used in air cylinders, each claiming some merits of its own.

In the earlier types of compressors ordinary leather flap valves were used, but these did not give the best results, owing to the large amount of leakage they permitted. A common type of valve that is still used to a large extent is the mushroom valve, fitted to a spindle and kept up to its seat by means of springs. Fig. 105 shows this type. These valves are opened automatically by the pressure of the air against the action of the springs, which must be of sufficient strength to close them against currents of air impinging on them. They are often difficult to keep in proper adjustment. If they are heavy, the springs must help to overcome their inertia; the latter are apt to get slack themselves through wear, and the valves then oscillate violently when they are open, which not only restricts the area of opening, but destroys them speedily together with the seats. Again, if the springs are too tight undue resistance is offered to the passage of the air when passing into the compressor, and this resistance is, of course, a dead loss of energy. This loss can be overcome to a certain extent by using valves of large area in order to keep the velocity of the air, while passing through them, as low as possible.

Riedler Valve.—In this valve no springs are used, the valves being operated by mechanical methods, and driven off the steam cylinder. A cam is fastened to the wrist-plate of the steam cylinder, and moves the rod attached to the air-valve gear by means of two steel rollers. Like the valves in the Riedler pump, these air valves are only closed mechanically. By using these valves the compressors may be driven at a very high speed, without injury to the valves, and no violent oscillation takes place, as in the mushroom type.

Losses in Compressing Air.—As already stated, only a very low efficiency is obtained by air-compressing machinery, and the various losses may be accounted for as follows:—

Heating of air during compression and cooling of compressor.
Loss due to air in clearance space in the cylinder.
Leakage at valves and piston.
Resistance of air in passing through inlet and delivery valves.
Loss due to friction in conveying the air from the receiver in pipes to the point of application underground, and also the friction of the air engine itself.

The two largest sources of loss are those due to heat and friction. The first often absorbs about 25 per cent. of the fuel expended, and the second about 20 per cent., the other losses being comparatively trifling compared to these. The loss due to heating will be readily understood from the fact that air, like any other gas, will expand when heated according to Charles's law; so that with the increase of volume due to the rise in temperature there will also be increased resistance to compression. To overcome heating during compression the following remedies, among others, have been suggested:—

To place the air and steam cylinders as widely apart as possible, so as to prevent the compressor being heated by the steam cylinder.

To place the compressing cylinders outside the engine-house, and simply protect them by a shed.

The air, before being admitted to compressor, should be reduced to zero by some freezing mixture.

Loss by Clearance.—In all cylinders a certain space must be left at each end of the stroke between the piston and the end of the cylinder, and the greater this clearance the larger will be the loss in efficiency. When the piston reaches the end of its stroke, the air which is entrapped in this space is compressed to a very high degree. As soon as the piston commences the return stroke, this air will begin to expand, but the inlet valves will remain closed until its pressure has become reduced below that of the atmosphere, *i.e.* the pressure of the air in the cylinder must be less than the pressure of the air outside of it before the suction valves will open.

To reduce loss from this source the clearance space should be as small as possible, or mechanical contrivances must be adopted. Thus a 'trick' passage may be made in the slide valve of a slide valve compressor, so that the air imprisoned in front of the piston can escape to the back of the piston and expand freely. The same object is obtained by having a pass-by groove at each end of the cylinder, so arranged that when the piston reaches the end of its stroke the entrapped air will pass to the other side of the piston, and allow the suction valves to open as soon as the piston begins the return stroke. In this way the loss due to clearance is reduced to a minimum, and no danger is incurred of damaging the cylinder covers.

All leaks in compressors, receivers, or pipes should, for the sake of economy, be strictly guarded against. Air leaks cause greater losses than steam leakage, and therefore no leakage should be allowed unless it is required to ventilate some place in the workings. Air at a pressure of 60 lbs. per sq. in. will have a velocity of 500 ft. per second. The great waste of power through leakage is therefore obvious.

All irregularities or quick bends should also be avoided as much as possible, as these materially increase the friction. One great source of loss from friction is that due to the machinery. As already pointed out, it often amounts to 25 per cent. or 30 per cent., but in well-designed plant it may be no greater than 12 or 15 per cent. The only way to reduce this loss is to secure accurate workmanship, well designed machinery, and efficient methods of lubrication.

High efficiency is often not all that is desirable, for if complex machinery is introduced, and skilled attention required, the benefit arising from increased efficiency may be more than counterbalanced by increased costs. What is wanted for mining purposes is a compression plant, simple in construction, with a low first cost, not easily put out of order, and yielding fairly efficient results.

Motors.—Other sources of loss are due to friction and leakage at the motor. The formation of ice in the latter and in the exhaust pipes is a difficulty nearly always encountered in using compressed air. The air, which always contains a certain amount of moisture when it reaches the motor, is at about the same temperature as the atmosphere, and when it enters, and is allowed to expand, the temperature will fall to such an extent that the moisture will immediately freeze. To obviate this, the exhaust passages should be made as large and straight as possible. Reheating the air would also overcome this difficulty, but this is not practicable underground.

Losses due to Leakage at Valves and Piston.—Losses of compressed air through valves and piston are often obstinate, owing to the difficulty of keeping the valves in proper adjustment, for, as already stated, the inlet valves may be set too tight and cause undue resistance to the air entering the cylinder. To avoid this, larger valves should be used. If they are badly constructed, the pressure of the air in the cylinder is often 9 or 10 lbs. higher than the pressure in the receiver, and, as a consequence, extra work has to be done to deliver the air against this increased pressure.

The leakage at the piston in air cylinders is often a serious loss. The moisture from the steam in a steam cylinder helps to keep joints tight, but with dry air there is no assistance obtained in this way. The only remedy for this defect is to provide the best workmanship in cylinders, and have the pistons finished to fit as truly as possible.

Loss due to Friction in Pipes.—In all liquids or gases delivered through pipes, there is a certain amount of friction between the fluids and the walls of the pipes. The loss owing to this friction will vary according to diameter of the pipes used, the pressure in them, irregularities, and the number of bends present, etc.

The size of pipes used should be as large as can conveniently be adopted, as this will tend to reduce the friction. In a column of pipes 1000 ft. long and 3 in. diameter, if the air has a velocity of 386 cub. ft. per minute, and a pressure of 60 lbs. per sq. in., the loss due to friction would be $3\frac{1}{3}$ lbs. In 6-in. diameter pipes of the same length and with the same pressure, the loss from friction would only amount to about $\frac{1}{9}$ lb.

The advantages of compressed air may be briefly stated as follows:—

Compressed air, being generated at the surface, is under the direct superintendence of the engineman.

It produces no increase in the temperature of the air in the workings, as is the case when steam is used.

There is not the same difficulty of dealing with the exhaust as there is with steam.

The exhaust air assists in the ventilation, and can even be used for ventilating in cases of emergency.

It can be easily applied to almost every kind of underground machinery, such as rock drills, coal-cutting machines, pumping and hauling engines, etc.

It is a safe motive power, and can be conveniently used in wet, dry, or fiery mines.

Electricity.—During the last decade a great increase has taken place in the application of electricity as a motive power, and it is growing in favour for certain classes of work.

The principal purposes for which it is adopted, and in which it has proved successful, are haulage, pumping, and lighting, and for such work it undoubtedly has many advantages over other motive agents, such as compressed air or steam. It is also used for shot-firing.

So far as results are obtainable, electricity does not seem to be very much cheaper, if, indeed, in many cases as cheap as steam used direct. It must also be borne in mind that there are certain dangers connected with the use of electricity which have not yet been overcome, and which still prevent its extensive use in a large number of mines. Electricity is capable of giving rise to a fire, and there is always the danger of an explosion from 'sparking' in a fiery mine.

There are, however, a large number of mines where these conditions do not exist and where fire-damp is only given off in small quantities which do not constitute any danger; in such mines, electricity may be used with the greatest advantage, economically and otherwise. The danger of explosion in fiery mines arises in two ways, viz., by 'sparking' at the motor, and by what is called 'short-circuiting' of the cables; that is, by faulty insulation, or by the breakage of the cable by a fall from the roof or some other cause.

Electric Mains or Cables.—To conduct an electric current from the place of generation (the dynamo) to where it is to be used (the motor), cables, or conductors, are required. Generally two, or for three-phase current, three such cables are required for a circuit, one to lead the current in and the other to lead it out, thus resembling a hydraulic motor with its supply and delivery pipes. Cables are usually of three types, viz., single-core cables; concentric or double-core cables; and three core cables. Opinions differ as to which is the best type for use underground.

Single-core cables have a single conductor placed in each. With this type two cables are required for the circuit. This class of cable is very largely used for colliery work, and is well suited to the rough wear and tear of underground work. They are easy to repair and joint, and are not so apt to 'short-circuit' as the other types, as they can be kept as far apart as the exigencies of the underground roads will allow. Concentric cables are those in which the intake and return conductors are both encased in a single cable, the conductors being separated from each other by insulating material. When the current has to be carried a long distance or down deep shafts, it is an advantage to use concentric cables, as it reduces the cost and there is less loss of potential. Concentric cables are, however, more troublesome to take branches and joints from, and more difficult to repair when damaged.

Three-core cables, as the name implies, are those in which three conductors are carried under the same covering. They are used for three-phase electric current, and, like concentric cables, are advantageous when the current has to be carried long distances, as they reduce loss of potential and save the carrying of three single parallel cables. All cables used underground should be well insulated to prevent risk of accident from shock by contact with 'live' wires. Bare or uninsulated cables should only be used when there is no such risk of contact.

Taped cables usually consist of a thin tape covering of either braided tape, steel tape, or wire tape. These cables should not be used for colliery work, unless for low voltages and where the workings are dry, as the taping is sure to get broken sooner or later and damage may be done.

Armoured Cables.—Cables for colliery work are frequently covered by armouring to protect them against mechanical injury from falls of roof, sides, etc. The armouring may consist of—

A single layer of galvanised iron wires, protected by jute compound.
A double layer as above, covered by jute compound.
A double layer of steel tape covered by jute compound.

It is recommended that the steel tape armouring should only be used for large cables, say over $\frac{19}{16}$ size.

Fixing Underground Cables.—Where cables have to be taken along main haulage roads or fixed in roadways where there is plenty of room, they may be attached to props or side timbers and supported so that they can be readily taken down for inspection or repair. A convenient and easy method of fixing cables is by leather strips, 1 in. to $1\frac{1}{2}$ in. wide, secured by flat-headed nails to the timbers every ten yards or so. In place of the leather strips some use pieces of tarred twine nailed to the props, and of sufficient strength to carry the cables. Both methods are quite efficient if well looked after, and if a fall of roof does take place the cables and fastening are carried to the ground with the débris with less injury than if the fastenings were stronger and of a more permanent character. Sometimes the cables are laid in wooden boxing, the boxing being afterwards run in full with pitch, which has the merit of keeping out damp and preventing mechanical injury to the cable. Wherever possible, cables should be taken along the intake airways, for if taken along the return they are more liable to wear and to damage from damp air and heat.

Shaft Cables.—Wherever cables have to be hung in the shaft, they should be highly insulated, preferably with vulcanised bitumen, and Mr Mountain recommends that they be further protected by wire armouring consisting of galvanised wires spirally wound round the bitumen insulation, the cables being afterwards heavily served with a good coating of jute compound. The armouring protects the cable from mechanical injury when fixed, and also enables them to be

lowered down the shaft without risk of damage to the insulation, because the copper conductors, especially in large cables, would be liable to stretch by reason of their own weight and so damage the bitumen insulation. In securing the cables in the shaft a variety of ways may be adopted. A common method is to fix them by wood cleats to the shaft sides or to buntons.

Junction Boxes.—The cables for underground roads should be coupled together by junction boxes, and such boxes should be used at all joints and branches. These junction boxes should also be used for testing purposes.

Voltages.—The pressure at which electrical mining plant is run varies greatly according to the design of the machine and the work for which it is intended. For colliery work with continuous current, the voltage used varies between 200 and 600 volts, 400 to 500 volts being a common pressure, and one which is not dangerously high when working with the continuous current type of machinery.

Mr W. C. Mountain, whose firm has had a very large experience in electric mining plant, considers 500 to 600 volts, both with continuous current and three-phase alternating current machinery, a most satisfactory voltage where the motors are placed not more than a mile from the generating plant, and he does not think that even with the three-phase machinery any great saving would be obtained by running at a higher voltage and putting in transformers. For underground motors, a voltage of 500 may be considered a practical working pressure, and is sufficient, provided the power required is not too great.

In continental mines, especially in Germany and Belgium, the three-phase system is almost universally employed, the voltages being 1000 to 3000, the current being transformed into a lower voltage, 500 to 600, by transformers at some suitable point underground, such as the pit bottom or at station at the end of a main haulage road. It should be remembered, however, that at many of these mines the whole of the operations on the surface and underground are carried on by electricity, necessitating large power. The shafts, too, are, as a rule, very deep, and under these conditions the use of three-phase electric current may be advantageous. At a number of British collieries, within recent years, three-phase plant has been laid down, but only in exceptional cases has a higher voltage than 500 or 600 volts been employed.

Generators or Dynamos.—Dynamos have been defined as machines for converting mechanical energy into electrical current or energy. They are usually divided into two classes, viz. :—

(*a*) Continuous current machines.
(*b*) Alternating current machines.

Each class comprises many different types, for technical descriptions of which the student should consult one of the many text-books on electrical engineering. Continuous-current machines are very largely

used for colliery work, and have been found very suitable for the varying conditions usually existing, and where the load is of a varying nature, as in hauling or coal-cutting.

Continuous-current dynamos are divided into three classes according to the method adopted in the winding, the three types being—

Series wound machines.
Shunt wound machines.
Compound wound machines.

Series Wound.—In this system the field-winding, *i.e.* the armature, the field-coil and the working circuit, are all in series receiving the same current, the current flowing from the positive brush through the field-coil windings, then through the external circuit and back to the negative brush. The whole of the current is sent through a coil consisting of a few windings of comparatively thick wire.

Shunt Wound.—In this type of winding a double path is open to the current. One part goes through the field-coil, which, in this instance, consists of a large number of windings of fine wire, of sufficient resistance and length to give the proper number of ampere-turns to fully excite the magnet, and is connected right across the brushes or poles to get the full pressure. The other part of the current flows through the external circuit, both currents joining at the negative brush before they return to the armature. The magnet coils act as a shunt to the main circuit. This type of dynamo is most suitable where a variable speed is required, such as, for instance, in main and tail rope haulage.

Compound Wound.—When the field-coil of a dynamo is wound with both a shunt and series coil of windings, it is called a compound dynamo. It will be seen that this is a combination of the two just described. The shunt coil consists of a large number of turns of fine wire calculated to give full potential at no load and with the magnet not fully excited, so that when the current increases in the external circuit it passes round the series coils, which are of thick wire, and increases the magnetism, and so raises the pressure to compensate for the drop in potential due to the resistances in the armature circuit. This arrangement of windings enables the dynamo to be self-regulating and give a constant E.M.F. with varying loads.

Motors.—A motor is a machine for converting electrical energy into mechanical energy. If we send a current through the armature of a dynamo whose magnetic field is excited, the armature will be put in motion. With the dynamo armature there will, however, take place, not only a single movement, but a permanent rotation. Owing to the action of the commutator, the current flows through all wires on one half of the armature which are under the influence of the north pole, in one direction, and through all wires which are under the influence of the south pole, in the opposite direction; hence as long as a current is sent through it, the armature will rotate. The machine now absorbs electrical and supplies mechanical energy.

In this case the machine is called an *electric motor*, which we may speak of simply as a *motor*, whereas a machine which produces current is called a dynamo or generator. As the construction of the motor is practically the same as the dynamo, we need not further describe it. There is one important difference, however, in the driving of the motor which may be pointed out; the armature always moves in an opposite direction to that of the dynamo armature. If to get a current the dynamo armature requires to turn to the right, then the motor armature will run towards the left.

Electrical Plant Failures.—Electric installations are not more liable to an excessive number of breakdowns than non-electric plant. With reasonable care in design, manufacture, and working, electric machinery can be, and has been, made as reliable, if not more so, than steam, hydraulic, compressed air, or oil-power plants. The first desideratum in colliery installations is to have ample power in the generating plant and also in the motors. Nearly all the trouble in the past has arisen owing to the plants being made too small for the work they are expected to perform.

The causes of accidents in electric plant are due to constructional defects; bad design and perishing insulation; bad workmanship; overloading; damp and dust, and oil and defective attention.

Electricity has, in England and Scotland, been chiefly applied to the operations of hauling and pumping. It is also used for coal-cutting machines, rock drills, and winding.

Compared with compressed air there can be no doubt that electricity gives a very much higher efficiency, often as much as 30 per cent. more. The efficiency of electrical machinery may vary from 40 to 60 per cent. In a paper read by Mr Robertson before the Institution of Civil Engineers, he states that an efficiency of 50 per cent. was obtained from the electrical haulage plant recently erected at Earnock Colliery, Hamilton. Comparing the indicated horse-power of the engine with the power developed on the hauling ropes the losses were—

	Per cent.
In engine, ...	22·00
,, belt and dynamo, . . .	8·00
,, cable, ...	12·50
,, motor and gear, . . .	7·50
Total loss, . .	50·00

At Haden Hill Colliery the electrical haulage plant gave 59 per cent. efficiency, the losses being—

	Per cent.
In dynamo, ...	9·00
,, cable, ...	8·00
,, motor, ...	9·00
,, gearing, ...	15·00
Total loss, . .	41·00

Another instance may, however, be given, where the tests made on a hauling plant gave an efficiency of 71 per cent.

Tests of Hauling Engines.

Distance of engine to hauling motor,	1400 yards
Energy imparted to dynamo,	22 H.P.
Work done by dynamo,	19·8 ,,
Loss of energy in dynamo,	2·4 ,,
,, ,, cables,	2·6 ,,
Energy imparted to motor,	17·2 ,,
Loss of energy in motor,	1·5 ,,
Work expended in raising coals and tubs,	12·1 ,,
,, ,, friction of moving tubs,	1·3 ,,
,, ,, haulage, gearing, and ropes,	2·3 ,,

The speed of the rope in this case was three miles per hour, and the total efficiency was 71·3 per cent., which is a very satisfactory result.

Comparing also the first cost of installing compressed air and electricity respectively, the latter is the cheaper. As a general rule an air-compressing plant will cost 20 per cent. to 30 per cent. more than an electrical plant capable of performing the same amount of work, while the cost of running the latter is 7½ per cent. to 10 per cent. less than that of running a compressed air installation of the same power.

Mr J. T. Forgie, in a paper read before the Mining Institute of Scotland, gives the following detailed costs of an electrical haulage plant erected at Dumbreck Colliery, Kilsyth * :—

Generating engine 20 in. diam. by 30 in. stroke complete,	£300	0	0
One dynamo, 50 horse-power,	374	0	0
Two motors, 20 horse-power each,	440	0	0
1750 feet of 19/17 cable,	93	0	0
Instruments, switches, etc.,	81	10	0
Labour, packing, carriage, etc.,	97	5	0
Two underground haulage arrangements,	400	0	0
Total cost,	£1785	15	0

The cost of a compressed air plant to do the same work may approximately be shown as follows :—

Steam engine, air cylinders, valves, etc.,	£1200	0	0
1750 feet air pipes 6 in. diameter,	180	16	0
Two air receivers,	70	0	0
Two haulage arrangements,	400	0	0
Labour in fixing pipes, etc.,	100	0	0
Sundries,	200	0	0
Total cost,	£2150	16	0

* *Trans. Inst. Min. Eng.*, vol. vii. p. 129.

Showing a difference in first cost of £365, 1s. in favour of the electrical plant, or a saving of approximately 23 per cent.

Pumping.—To no branch of mining has electricity been more successfully applied than that of pumping, and there can be no doubt of its suitability for such purposes, owing to the small space occupied by the driving motor compared with a steam engine, and by cables compared with steam or air pipes, together with the ease with which the former may be carried to any part of the workings.

The cost of the cable will be only about one-half that of pipes.

Electrical pumping installations differ very widely in efficiency, a great deal depending on the suitability of the motor for the class of work required. In a paper read before the Institution of Civil Engineers by L. B. and C. W. Atkinson, describing an electrical pumping plant, the gross efficiency was given at 49 per cent., the generating engine being placed 1200 yds. distant from the pumps. Comparing the I.H.P. of the engine, which was 31·75, with the volume of water delivered, the losses were distributed as follows:—

	Per cent.
Loss in generating engine and in belts,	31·50
,, dynamo,	6·05
,, cable,	5·36
,, motor,	4·72
,, pump and gear,	3·15
Total loss,	50·78

A very large installation of electrical pumping plant has recently been put down at Arniston Colliery, near Edinburgh, the following particulars concerning which may be of interest.

Gore Pit.—One set of three-throw pumps, ram 11 in. diameter by 18 in. stroke, to deliver 500 gallons per minute, against a head of 678 ft., at an approximate speed of thirty revolutions per minute.

Emily Pit.—One set of pumps exactly similar to the above, to deliver 500 gallons per minute against a head of 250 ft. through 3175 ft. of cast-iron pipes 10 in. diameter. Three sets of pumps, 6 in. diameter by 9 in. stroke, in the dook, each set delivering 100 gallons per minute against a head of 450 ft. These pumps deliver through 1200 ft. of cast-iron pipes 6 in. diameter.

Motors.—For driving the large high-lift pump, which delivers 500 gallons against a head of 678 ft., two 80 horse-power motors are used with an approximate speed of 450 revolutions per minute. The second set of pumps in the Emily Pit are driven by a single motor of 80 horse-power, at a speed of 450 revolutions per minute. The three sets of pumps in the dip workings are driven by a motor giving 25 effective horse-power at a speed of 250 revolutions per minute. The total cost of plant, exclusive of the power stations, was £12,000.

Generating Plant.—The power was furnished by two compound horizontal engines, having cylinders 16¾ in. and 26½ in. diameter by

36 in. stroke each, and working at a speed of eighty-four revolutions per minute. Each engine is capable of developing 350 horse-power, with a steam pressure of 120 lbs. per sq. in.

Dynamos.—There are two dynamos, fitted with drum armatures, each dynamo being constructed to yield the following:—

Total watts,	200,000
Amperes,	363
Volts,	550
Approximate revolutions per minute,	400

The working of this large and expensive installation will be watched with much interest by those engaged in mining.

It should be stated that no fire-damp is ever encountered in the workings, the whole being worked with naked lights; no danger is therefore to be apprehended from gas explosions.

One of the great advantages claimed for the electrical transmission of power is, that a dynamo or a motor is self-regulating, *i.e.* the dynamo only requires sufficient power to drive it, to enable it to accomplish the work which it is called upon to perform, and a motor only requires current sufficient for the same purpose. It is of the greatest importance that the dynamo or motor should be large enough for the work. The best results are only obtainable with a fair surplus of power in the steam engine. The advantages and disadvantages of electricity as a motive power may be briefly stated thus:—

Advantages:—

The great facility with which it can be used in any part of the workings, and a motor put down wherever required, for driving a pump or haulage system.

The small amount of space occupied by the motor, while, owing to the high speed at which it works, a large amount of power can be applied from quite a small pulley by belting.

It does away with the danger of ropes or pipes in the shaft, and avoids the complication of pulleys and ropes at the pit-head and pit-bottom.

The cables can be readily fixed and taken round curves; there are no joints to be affected (as in pipes) by vibrations or shocks, and the space occupied by conductors is very small.

The surface plant can be placed any distance from the shaft and not necessarily in line with the latter.

Higher efficiency can be obtained than with compressed air or steam when used underground.

Disadvantages:—

The danger of fire, either igniting fire-damp or setting fire to screen-cloth or brattice-wood, owing to sparking at the motor or 'short-circuiting' in the cables.

Electric machinery is easily damaged and thrown out of order, and often requires skilled men to repair it.

Its unsuitability in damp and dirty workings, as the cables and beltings soon suffer injury under such conditions.

The first cost is much higher than for ordinary haulage engines driven by steam.

The following table shows the comparison in the cost of transmitting power by the various systems in use * :—

System employed.	330 ft.	1640 ft	3280 ft.	16,400 ft.	32,810 ft.	65,250 ft.	Motor.
	Pence.	Pence.	Pence.	Pence.	Pence.	Pence.	
Electricity, . .	1·80	1·84	1·94	2·27	2·84	4·61	Steam.
Water pressure, .	2·27	2·36	2·61	4·03	5·79	9·57	
Compressed air, .	3·17	3·23	3·33	4·05	5 25	6·39	
Ropes, . .	1·26	1·49	1·58	2·88	4·99	12·68	
Electricity, . .	0·46	0·48	0·53	0·62	0·68	1·15	Hydraulic.
Water pressure, .	0·46	0·55	0·62	1·33	3·23	3·54	
Compressed air, .	0·72	0·80	0·84	1·21	1·86	3·36	
Ropes, . .	0·25	0·27	0·30	0·80	1·42	3·39	

The cost as given above is for the transmission of 100 horse-power for the distance tabulated, from which it would seem that transmission by wire ropes is the cheaper up to 3000 ft., and electricity for longer distances.

Electrical Terms.—The volume of an electrical current is measured in amperes in the same way as the speed of air in pipes is measured in cubic feet per minute or second, or that of water in gallons or feet per second. The tension or pressure of an electrical current is measured in volts, and corresponds to the measurement of steam pressure by pounds per square inch. An electrical horse-power equals 746 watts, a watt being the power conveyed by a current of one ampere at a pressure of one volt in a second of time; or, in other words, when a current of one ampere passes through a resistance of one ohm in one second. The resistance of any conductor to the passage of electricity is measured in ohms.

The difference of pressure of electrical energy at varying points is usually spoken of as electromotive force (written E.M.F.), and is measured in volts.

Let E = electromotive force in volts,
R = resistance in ohms,
C = the current in coulombs,

$$\text{then } E = R \times C \text{ or } C = \frac{E}{R}$$

Work done in mechanics is usually measured in foot lbs.

In electricity, however, the watt is the unit of work (= volt × amp.)

$$\text{H.P. in mechanics} = \frac{\text{foot lbs.}}{33,000} = 746 \text{ watts.}$$

$$\text{H.P. in electricity} = \frac{\text{watts}}{746} = \frac{\text{volts} \times \text{amperes.}}{746}$$

* *Trans. Inst. Civil Engs.*

Power given out by a current of	1 ampere	at a tension of	1 volt	= 1 watt	
,, ,, ,,	2 amperes	,, ,,	2 volts	= 4 watts	
,, ,, ,,	10 ,,	,, ,,	2 ,,	= 20 ,,	
,, ,, ,,	74·6 ,,	,, ,,	10 ,,	= 746 ,,	= 1 H.P.
,, ,, ,,	7·46 ,,	,, ,,	100 ,,	= 746 ,,	= 1 H.P.

That is, a current of 74·6 amperes at a tension of ten volts, or one of 7·46 amperes at a tension of 100 volts, is equal to one electrical horse-power.

A wire of a given size will permit the flow of a given number of amperes proportional to its diameter. For instance, a conductor of 19 wires, each wire being 16 B.W.G., is suitable for the flow of about 50 amperes for a distance of 1000 yds., with a loss in the conductor of, roughly, 10 per cent. If the voltage of the above current be 500, then the horse-power will be $= \frac{500 \times 50}{746} = 33{\cdot}5$.

For large current and small pressure or voltage, large conductors must be used, which are expensive. For small current and high voltage small conductors may, however, be employed, which is more economical, although they increase the danger arising from sparking, short circuits, etc. For underground work voltages exceeding 500 are rarely used, owing to the danger of sparking and of shock to men coming in contact with the cables. For electric lighting with incandescent lamps, a voltage of 50 to 100 volts is usually employed. An incandescent lamp of 20 candle-power at 100 volts takes about 0·6 ampere, *i.e.* $100 \times {\cdot}6 = 60$ watts, or 3 watts per candle-power.*

* For further details of the application of electrical power in mines the reader is referred to *Electrical Practice in Collieries*, by Daniel Burns, M.Inst.M.E. London, second edition, 1905.

CHAPTER VIII.

MODES OF WORKING.

Choice of Methods.—The main object in any system of working is to extract as much of the coal as possible, with the maximum of economy and safety, and in longwall, the coal left in ought not to exceed 5 per cent. to 7 per cent. of the total quantity in the seam.

The two principal methods of working are pillar and stall (stoop and room in Scotland), and longwall, all other systems being simply modifications or combinations of these two.

The method of working any seam naturally depends on local circumstances in each individual colliery, and, speaking generally, varies according to the thickness of the coal. Seams of 4 ft. and upwards are usually worked by pillar and stall; seams having a thickness of less than 4 ft. are usually worked by longwall. There are, however, exceptions to this rule.

Besides the thickness of the seam, the mode of working will depend on other circumstances, such as—

The inclination of the strata and the nature of roof and pavement.
The depth of the seam from the surface.
The chemical and physical properties of the coal.
The natural cleavage of the coal and that of the rocks forming the roof.
The presence or absence of water.
The vicinity of other seams or of other workings which should not be interfered with.
The number of dykes and dislocations in the field to be worked.

Longwall.—In this system the whole of the mineral is usually extracted in one operation. In some cases a modification of the system is adopted, and pillars are left along the main haulage roads to help to maintain them, but it is very doubtful if this is an advantage, as it generally entails the loss of a large percentage of coal, which might otherwise have been got, and, as far as the security of the road is concerned, it would seem, from recent investigation, to do more harm than good.

To work a seam to the best advantage by this method, there must be—

A fairly good roof and pavement.
The roof and pavement should be free from water.
The seam must be neither too thick nor too highly inclined.
The coal itself should neither be too soft nor too friable.

A very soft, friable seam with a hard roof and pavement is usually unsuitable for longwall, no matter what its thickness may be.

Thick seams, worked on the longwall system, are generally more dangerous, from falls of roof and sides, owing to the roads being imperfectly built for want of proper stowage; they are also more expensive as regards timber, but, as a rule, a larger percentage of round coal is got than by pillar and stall system. The workings are generally laid out in a regular manner, after the seam has been opened

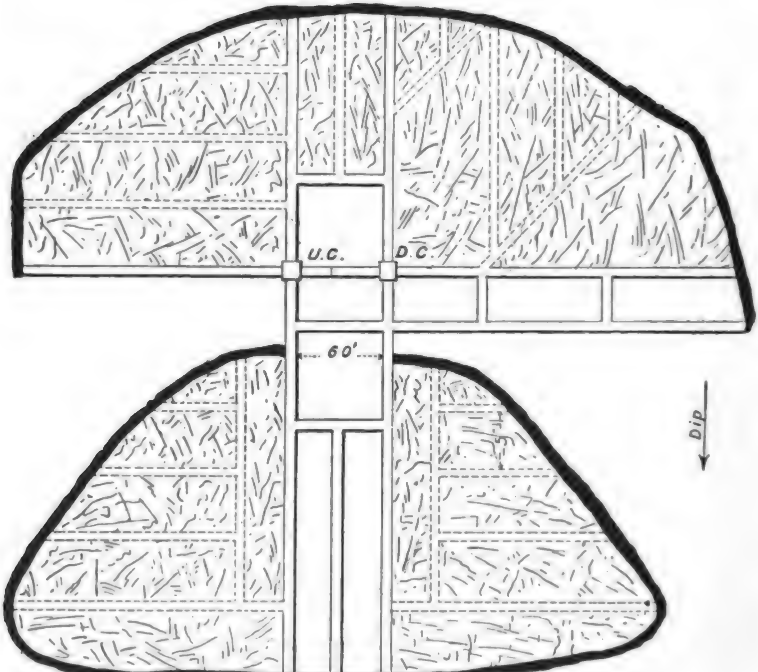


FIG. 106.—Longwall working.

out, the main haulage or drawing roads being set out to the full dip or rise, with side or branch roads at right angles to them. Fig. 106 shows a method of laying out the workings in which the main roads are carried to the full dip and rise, and three parallel roads are used, the two outside roads being utilised as intake airways, and the centre one as the return.

An important factor in determining the direction of the roads is the jointing or clearance of the coal. These joints or cleats cross one another at right angles, but there is always one direction along which the coal yields more easily than in any other, known as the main

cleat or back. As a general rule, it is better to drive roads at right angles to the line of main cleat ('on plane'), and not to set off a distance in advance, extending beyond the next main cleat. Sometimes, however, if the seam is soft and friable, it is better to work the roads parallel to the main cleat, and it also suits the inclination of the roads to sometimes work along the bedding. If the cleats are good and the coal soft, it is often best to drive the places or walls 'half on end' (*i.e.* at an angle of 45° to the line of main cleat). Joints in the roof sometimes coincide with the cleats in the coal, when, if the walls are driven parallel to them, the roof becomes bad and dangerous, and repair of the roads difficult and expensive.

Length of Walls.—The length of wall depends on the thickness of the seam and the amount of material at disposal for 'packs.' For a seam 4 ft. thick, with a good roof, 12 to 15 yds. is quite long enough; for a 3½ ft. seam, 15 to 20 yds. is sufficient; and for seams 1½ ft. to 2 ft. thick, the walls may be 20 to 25 yds. in length. In some districts with moderately thick seams, 4 to 5 ft., the walls are often as much as 40 to 60 yds. in length.

In thin seams the walls ought to be long enough to hold all the débris, and the longer the walls the less will be the cost for ripping. If, however, the walls are too long (in thin seams) the coal is much injured by breakage, in the process of throwing it two or three times along the wall to the road-head or gate-road.

Width of Roads.—The width of roads varies from 6 to 10 ft., according to the depth, nature of roof and floor, and thickness of seam. They should be at least 7 ft. wide and 5½ ft. high, as narrow roads give much trouble, owing to the packs getting squeezed out and catching on the tubs.

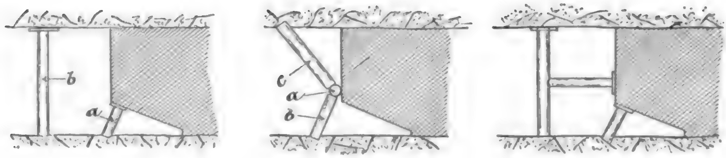
Size of 'Buildings' or Packs.—The size of packs along the face varies from 6 to 12 ft., according to depth, but no pack ought to be less than 6 ft. along the face. The size naturally depends largely on the amount of rubbish available in the workings, but a good-sized pack for a 4 ft. seam at a depth of 120 fms. would be 12 ft. along the face. The size of pack will depend largely on the amount of refuse got in the working of the seam, and in its thickness. In seams where there is a large amount of rubbish got in working and in ripping the roads, the whole of the space from which the coal has been extracted may be completely stowed or packed.

The side or stall roads should not be kept too long in use, as they often become dangerous and require frequent repairs to keep them open.

Level or slope roads should be set away every 50 fms., or, where the roof is bad, every 35 or 40 fms. The cost of upkeep may be lessened by ripping anew every tenth or twelfth side road, after complete subsidence, and converting them into main drawing roads. By this means rails and sleepers are economised and ventilation is facilitated. The re-arrangement certainly entails additional expense, as it will cost 6s. to 7s. 6d. per fathom for back ripping these roads, but the gain will

in other ways more than compensate for this outlay. The cost for ripping in ordinary workings varies from 3d. to 6d. per ton of coal produced, but, as a general rule, it should never exceed 6d. per ton.

The working places ought to be carefully propped and no more timber left in the waste (goaf) than can be avoided, as it keeps the roof from subsiding properly, and needlessly increases the cost. The coal ought also to be properly spragged with 'holing props' to get the full benefit of the 'weight' when it comes on the coal.



(1) (2) (3)

FIG. 107.—Spragging.

Fig. 107 shows the methods of using the holing props. When the coal is soft and difficult to keep up while it is being holed, the method shown in fig. 107 (2) or (3) is adopted, while, in ordinary circumstances, the method shown in fig. 107 (1) is used. This simply consists of setting up a series of short props or sprags *a*, either with or without a lid, along the face at the edge of the holing. When the coal is soft a horizontal prop (fig. 107 (2), *a*) is placed parallel to and against the coal, a short prop, *b*, supports it from the floor, while another prop, *c*, set at an angle to the roof, helps to keep it in position. The same result may be attained by using the method shown in fig. 107 (3). The holing props should be set up at intervals of not more than 6 ft. apart, and this rule ought to be strictly enforced, as a great many accidents from falls of coal are due to the neglect of using holing props (see C. M. R. Act as to rules for spragging the coal). By properly spragging the coal a larger amount of round is obtained and a smaller percentage of dross than when this precaution is neglected.

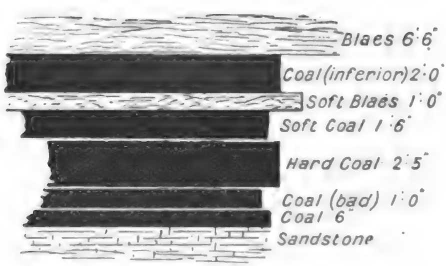


FIG. 108.—Nottingham 'Top Hard' seam.

In some districts where longwall is practised, the walls are made as long as possible, 25 to 60 or even 70 yds., and a road laid along the face to take in the tubs. This is a decided advantage where a large area of coal can be opened out, as it saves expenditure for ripping, and few roads require to be kept open for a given length of face.

In the 'Top Hard' seam at Nottingham, of which a section is given, fig. 108, the method of working is longwall, the walls being 25 to 30 yds. in length. The 'holing' is done in the soft blaes,* and sprags are put in every 6 ft. The places are worked on 'end,' and no blasting is required; when it is worked on 'plane' the coal is more crushed. The rubbish from the walls and ripping is sufficient to pack the whole of the waste, and a pack is put in when there is 4 ft. of ripping available. Two rows of wood, 5 in. diameter, are kept between the pack and the coal, the back row props being placed at a distance of not more than 6 ft. apart. The tram road is laid between these two rows of props (fig. 109). The coal having been holed, the whole length of the wall and the rails are lifted, a 'cut' is taken simultaneously right and left along the wall, the rails being laid down anew from either side at the same time, and a new pack of 5 ft. is put in (at X).

The props of the back row are drawn and set up at the face again.

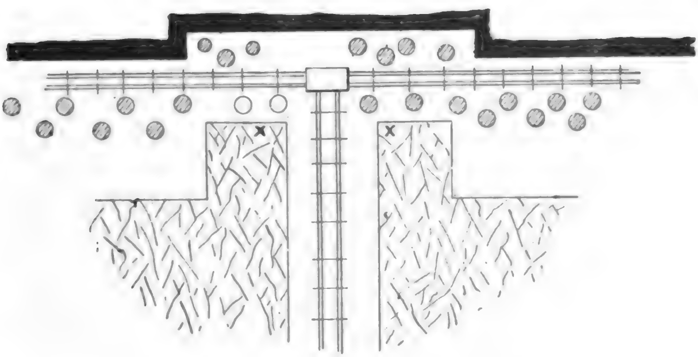


FIG. 109.—Plan of wall showing tram road.

The branch roads are cut off every 50 fms. by new slopes, and the top coal is taken down by these roads. In the main roads the $6\frac{1}{2}$ ft. of shale is also taken down, a good high road resulting. The output per man averages four tons per day.

In the Main seam at the same colliery the working is similar, but the walls here are 60 yds. long, and the seam is worked on 'end' as before, no blasting being required. The under coal and fireclay are holed to a depth of 5 or 6 ft., and sprags put in.

The rubbish got from ripping and holing is not sufficient to fill the whole of the goaf, and packs 6 ft. thick and 9 ft. apart are put in at intervals along the wall, while two, and sometimes three, rows of props of 6 in. or 8 in. diameter wood are set up, 6 ft. apart, with a distance of 3 ft. between the rows; the back props being drawn and

* Argillaceous shale.

shifted forward as packing proceeds. The tubs are taken along the face between the inner row of props and the coal. Where the roof is tender, straps composed of old rails are put up between the coal and the props in the nearest row, the inner end being wedged into the coal.

The 'Drumgray' seam in Lanarkshire varies in thickness from 1½ ft. to 2½ ft., and is worked longwall. The seam in some districts lies very level, and the general mode of working is to set off roads in the direction of the rise and drive branches, right and left, the walls being 15 to 20 yds. long. The coal is hauled by boys along the face in small bogies to the 'road-head,' and there filled into the tubs. The roads are ripped to a height of 5 or 5½ ft., and made 7½ ft. wide, the rubbish got from the ripping and holing being sufficient to pack the whole of the goaf. The ripping of the roads costs 7s. 6d. to 10s. 6d. per fathom, and frequent repairs are necessary to keep them in order.

At Westrigg Colliery, near Bathgate, the Drumgray seam is only 14 to 17 in. thick, with a hard rock roof. It lies flat and does not contain much water, although the other seams at this colliery are very wet. Owing to the roof being strong, very little timber is required, but the roads soon get very low. The holing is done in the coal, the floor being hard shale. Each man keeps his own place in order, and does the ripping of his own road, the price paid in 1898 being about 4s. per ton of coal got. This price includes the ripping of roads and the drawing of the coal. The output per working place is about three tons per shift of nine hours. To work so thin a seam would be almost an impossibility with a bad roof, as the cost of upkeep would be too great; but, in this instance, very little extra expenditure is required, the fireman alone being generally able to examine and keep fifteen to twenty places going without much trouble.

Sand or Water Pack.—In British mines where seams are worked on the pillar and stall system, it is not usual to pack the waste ground when the pillars are extracted, the timber being usually withdrawn and the roof allowed to collapse. If the seam is thick and situated at no great depth much damage may be done to the surface when the pillars are extracted, especially in the vicinity of towns, where the buildings may be partially wrecked, of which many instances have occurred throughout Britain. To obviate this the goaf may be packed with sand or fine débris run in with water through a series of bore-holes from the surface. This process was first introduced in America in connection with the working of the thick anthracite seams, which in many places lie at no great depth from the surface. The practice there is to utilise the fine refuse and sludge from the screening and coal-washing plant, which is sluiced with water into the waste workings.

The system has also been successfully applied in Germany, where in some districts, such as Silesia, thick seams, 20 to 30 ft., have to be worked, and where previous to the introduction of the sand-pack a large quantity of the coal was lost in the removal of the pillars,

and much damage done to the surface. The chief material used for this purpose in the Silesian mines is sand, although other material, such as coal-dust, cinders, clay, refuse from coal washeries, ground stones and bricks, are also used. The fine débris is carried down the shafts in pipes, and thence to the required position in the workings, by a stream of water, the water being allowed to filter through the pack and then pumped to the surface. "At the space to be packed underground, the delivery pipe is raised as close to the roof as practicable at the upper end of the open space, and the openings, except that for the delivery pipe, are closed by dams. Water alone is then allowed to flow through the pipes for some time so as to clear them thoroughly, and afterwards the tipping of the material into them, which is done through a funnel, is commenced. The water, after depositing its load, filters away through a dam placed at the bottom end of the pillar, eventually reaching clearing tanks, whence it is pumped to the surface. Meanwhile the material is gradually depositing and being tightened by its own weight, until only the small space round the mouth of the delivery pipe remains unfilled; the flow is then turned off in this direction, and the same operation repeated with it by means of branch pipes somewhere else. It is advisable to have telephonic communication between the tipping and delivery ends of the pipes. After giving the packing a day or two to dry, the dams are removed, and the coal in the neighbourhood can be worked: the packing remains quite tight and firm, and resembles a natural stratified bed." * The cost at one colliery in Germany for packing the waste in this way was about 6d. per ton of coal won, this including the cost of pumping, pipes, interest on capital—everything, in fact, except the sand and water, which were got free. There are few districts in Britain where sand could be had free in sufficient quantities to carry out this system successfully, but at many collieries there are large heaps of refuse which might be utilised for the purpose. The principal advantages claimed for this method of packing are: (*a*) a larger quantity of coal can be won; (*b*) little damage is done to the surface; (*c*) less timber is required.

To secure success in working longwall it is necessary that—

The places or walls should be kept going regularly, and fully equipped with the full complement of men, otherwise some of the places will fall behind and will cause trouble with the ventilation, and in other ways.

The line of face should be kept as even as possible, and unless the seam is very highly inclined, not worked zigzag, as the point or 'nose' is sure to get crushed, the 'ribside' will not have sufficient weight, and very often the wall will get closed at this point altogether, hindering ventilation and causing trouble and expense in 'winning-out' again.

No portion of the face should be in advance of the rest further than a single 'cut,' as this makes the coal more difficult to get, and more dross ('slack') will be made by the increased 'shearing' and 'holing.'

The places should be carefully propped, while no timber should be left in

* *Trans. I. M. Engs.*, vol. xxviii. p. 325.

the waste that can be avoided, as it keeps the roof from subsiding properly, besides increasing timber costs.

The holed coal ought to be carefully spragged by holing props or 'gibs,' to get the full benefit of the weight when it comes on.

The ripping in each road should always be kept well forward and the buildings (packs) well and tightly laid. If they are loose, much trouble is often caused by their being pushed out into the roads when the weight comes on them.

Advantages of Longwall.—The advantages to be derived from working any seam by longwall may be briefly stated thus:—

The coal is generally extracted with only 5 per cent. to $7\frac{1}{2}$ per cent. loss, resulting from places 'closing' and requiring ribs of coal left in, and from bottom pillars required.

As the 'shearing' is confined to one or two places or main roads, there is a considerable saving in that part of the work, and, therefore, better coal at lower cost is obtainable than where much shearing is required.

The coal is easier to work, and the working price is cheaper, as a rule, than in bord and pillar.

The 'weight' often reduces the labour of getting the coal, for, if properly taken advantage of, it helps to bring down the coal after being 'holed,' thus saving expense in blasting and giving from 10 to 15 per cent. more round coal than pillar and stall working.

Ease in ventilation, small cost for bratticing, and comparatively short distance for haulage for a given output.

More men can be employed in a given area, and, therefore, a larger output can be got than in pillar and stall working.

Fewer roads require to be kept open for a given output, and there is a consequent saving in rails, sleepers, timber, etc.

With thin seams and where the conditions are suitable, coal-cutting machines can be used.

The disadvantages are:—

The roads are more difficult to keep open than in pillar and stall, especially if the roof and floor are wet, and the latter tends to creep.*

Unless the work proceeds regularly, roads and faces are difficult to keep open, and the ventilation is hindered.

Dykes and dislocations are more difficult to deal with than in pillar and stall working, and cause much trouble with the roof and sides.

Longwall working is unsuitable for thick seams with little rubbish available for packs in the wall.

It would seem from the Inspectors of Mines' Reports that there is practically no difference between longwall and pillar and stall, so far as safety and the number of accidents are concerned.

Pillar and Stall, Bord and Pillar, or Stoop and Room.—This mode of working, with its numerous modifications, may be said to be the only other method of working a seam unsuitable for longwall. Seams from 4 ft. thick and upwards may be worked by pillar and stall, and some even thinner than 3 ft., where the pavement is soft

* The word 'creep' is usually confined to the slow rising of floors, which sometimes, owing to the pressure of the walls on either side, become more and more convex, and sometimes even block up the road or render it impassable.

fireclay, a portion being lifted, as in some districts of Cumberland. If a seam situated below another working containing water has to be worked, it is often the best method to adopt, no matter what thickness the seam may be. The system, however, is best suited for thick or moderately thick seams, with no available débris. In seams of a soft or friable nature with a rock roof it is likewise, as a rule, the best method of working.

Pillar and stall working is divided into two distinct operations:

> Driving places in the solid coal, and dividing the area of coal to be worked into square or rectangular blocks or pillars by narrow places called 'stalls,' 'rooms,' or 'drifts.'
>
> Extracting the pillars and allowing the roof to fall in and fill up the space left. This is usually the most difficult and important part of the work.

The accompanying illustration (fig. 110) shows the pillars or stoops, and the stalls, bords, or rooms.

Size of Bottom Pillar.—The first important point to be considered is the size of the pillar to be left at the bottom of the shaft to protect the surface buildings and the shaft itself from damage. The size of shaft pillars is given differently by different authorities. A good-sized pillar is one 40 yds. square for a depth of 50 fms., and the size should increase by 5 yds. for every 10 fms. increase in depth of shaft; *i.e.* the size of pillar for 60 fms. would be 45 yds. square, and for 70 fms. deep 50 yds. square, etc.

Another plan, which is much adopted in Scotland, is to draw a line enclosing all the surface buildings, such as engine-houses, fans, screens, etc., that it is necessary to protect, and make the shaft pillar of such a size that solid coal will be left in all round this line for a distance equal to one-third of the depth to the seam.

The size of bottom pillar may also be calculated from the formula $R = \frac{\sqrt{3d} \times \sqrt[3]{t}}{0 \cdot 8}$, where R = radius of pillar in yards, D = depth from surface in yards, T = thickness of seam in feet.

André gives the following sizes as suitable for shaft pillars:—

Up to 150	yds. deep,	pillar	35	yds.	square.
,, 175	,,	,,	40	,,	,,
,, 200	,,	,,	45	,,	,,

the size increasing 5 yds. for every 25 yds. increase in depth.

The writer's own experience leads him to believe that safety is secured by making the shaft pillar area equal in length in yards to the depth of the shaft in fathoms. For a depth of 100 fms. the shaft pillar should be 100 yds. square; for 120 fms. 120 yds. square, etc. The size of pillar, however, must depend on the nature of the coal, the roof and the floor, as well as the inclination of the seam and the extent and nature of the surface buildings. It must also be remembered that bottom pillars should have an excess rather than a

deficiency of coal, as they are frequently cut up later for new haulage roads, etc. In steeply-inclined workings at least $\frac{2}{3}$ of the pillar

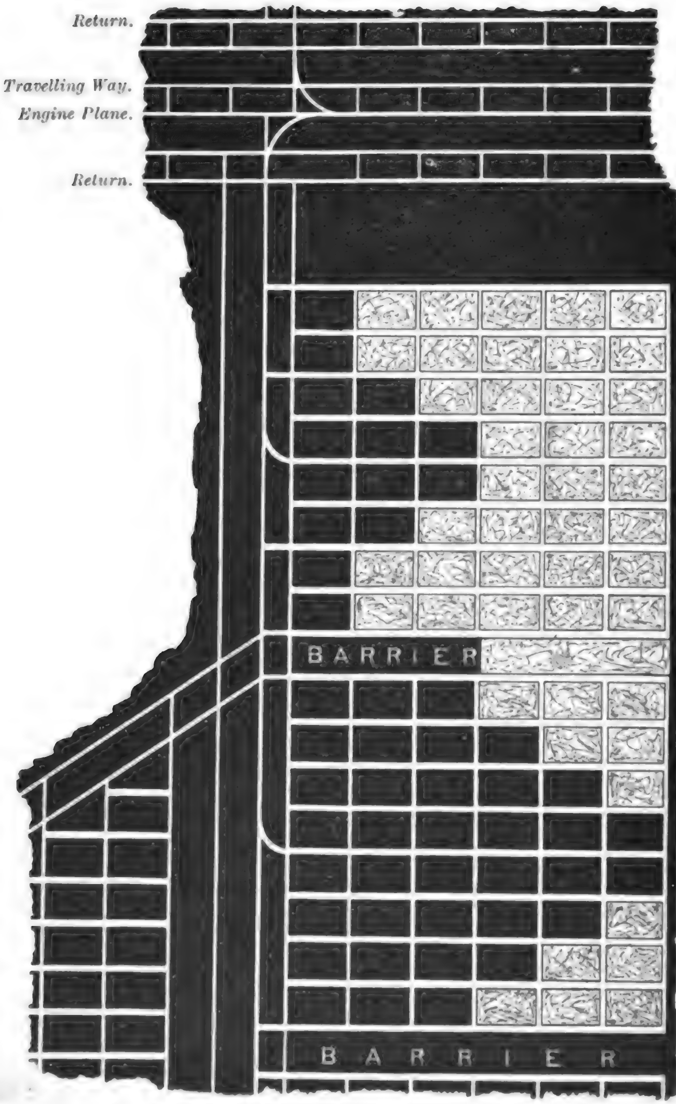


FIG. 110.—Bord and pillar system.

should be left on the rise side of the shaft, as the 'weight' always tends downhill.

The size of pillars in the ordinary working varies very much in different localities and in different seams. In the Hamilton district of Lanarkshire, in the Ell coal, which averages about 5 ft. thick and is at a depth of 100 fms. from the surface, a common size is 30 yds. × 20 yds., while for the Splint coal in the same district, which is usually about 6 ft. thick and lies about 20 fms. or so deeper than Ell, the pillar is often 30 yds. × 30 yds. or 40 yds. × 30 yds. The pillars left in the first working are much larger now than was formerly the case; often in the first working only 8 to 10 per cent. being taken out, and 90 to 92 per cent. left in the pillars, according to circumstances. At Milnwood Colliery, Bellshill, in the Splint coal, only 8½ per cent. was taken out and 9½ per cent. left in the pillars.

For determining the size of pillars the following rule may be used: Allow 6 sq. yds. of pillar per fathom depth for a 4-feet seam, 8 sq. yds. per fathom for a 5-ft. seam, and 10 sq. yds per fathom for a 6 ft. seam, etc.; multiply this allowance by the number of fathoms in depth, and extract the square root of the result. In the case of a 4-ft. seam at a depth of 120 fms., we have $120 \times 6 = 720$; $\sqrt{720} = 26{\cdot}8$ yds. square.

The size of pillar may also be ascertained approximately by the formula, $S = \sqrt{\frac{D}{50}} + 22$, where S = size of pillar in yards, D = depth of seam in yards. Taking all things into consideration, pillars as large as possible should be left in during the first working.

If roof and floor are both soft and only small pillars are left, 'creep' will occur * (fig. 111), and a large amount of the coal may be entirely lost. 'Robbing' the pillars in an irregular manner may also bring on 'creep,' and large areas of valuable coal have been lost in this way. Pillars sufficient to prevent creep in the first working may be quite incapable of doing so when the work of extracting has commenced. Again, if the roof is bad, the floor hard, and small pillars only have been left, then when a crush comes on the roof will 'spill' or 'ride' over the pillars, necessitating a large amount of 'redding' when stooping is going on, and much of the coal will be crushed into small, of much less value.

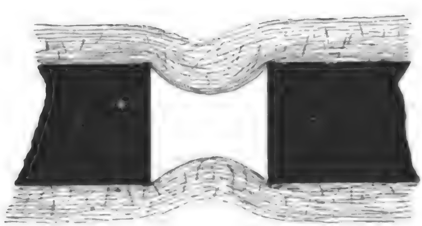


FIG. 111.—Showing creep and crush.

* See note, page 144.

If both roof and pavement are hard and small pillars are left, the coal will be much crushed, and a large percentage of dross will be got. When this takes place it is called 'thrust' or crush.

Another point to be taken into consideration in fixing the size of the pillars is the amount of output required daily. Generally speaking, small square pillars are most favourable for a large output, as the stalls are numerous, and help to win out new places speedily.

Direction of Pillars.—Pillars should, as a general rule, be worked lengthwise to the rise of the seam, unless the latter becomes very steep, when it is found more economical to make the long side of the pillars at right angles to the inclination, and so have a larger amount of coal to work from on the level course (fig. 112). The most important point is to have the pillars running parallel with the main cleat, as this secures the best coal at the least cost.

Width of Stalls or Rooms.—This will depend almost entirely on

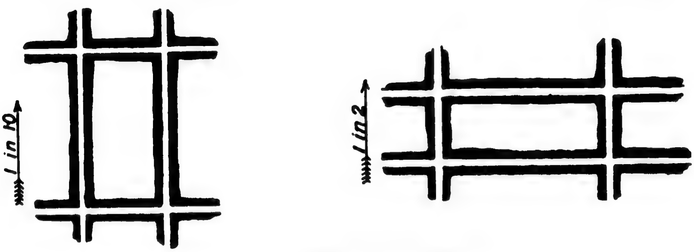


FIG. 112.—Direction of pillars.

the nature of the roof, but if the latter is good, the stalls should be driven as wide as possible, consistent with the safety of the men.

Wide stalls give a much larger percentage extraction of round coal than narrow ones, which is a very important point, except where the coal is to be coked, in which case it does not so much matter about small coal being produced. With a stall 6 ft. wide there will be an average of 25 per cent. dross (slack); with a stall in the same seam 15 ft. wide, the average would be only about 15 per cent.

In Scotland, the 'rooms' and 'ends' are usually the same width, varying from 9 to 12 ft.; in the splint coal of Lanarkshire, which has a rather soft roof, the stalls are usually 8 or 9 ft. wide, but in other seams where stoop and room is practised 12 ft. is a common width. In the North of England the 'bords' or stalls are narrow, being only 8 to 12 ft., while the 'throughers,' or places at right angles to the 'bords,' are often as many yards wide. Again, in other districts, it is the practice to open out both 'bords' and 'throughers' narrow (6 to 9 ft. wide for a short distance), and then to increase the width to 8

or 9 yds., such methods being common in Cumberland, for instance. No hard and fast rule can be laid down for the width of openings, the main requirement being to keep the stalls of such a width that the minimum of dross or slack will be obtained with the maximum of safety.

Extracting the Pillars.—This is the most important and dangerous part of the work in pillar and stall working, and great care should be exercised in carrying it out. The chief point to aim at is to get out the coal as quickly as possible, without endangering the men or losing a whole 'lift' ('judd') of timber for the sake of a tub or two of coal. On the other hand, no coal ought to be left in that can

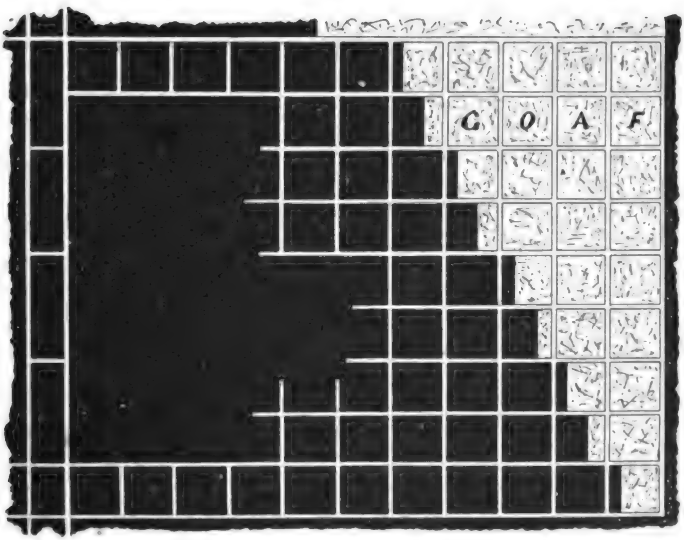


FIG. 113.—Extracting pillars.

possibly be extracted. To satisfy both conditions, the pillars ought to be worked out in a regular manner, beginning next the goaf or boundary, and working forward in regular order (fig. 113).

Too many pillars should not be removed at once. As soon as a section of solid coal has been driven through, it ought to be extracted as quickly as possible, otherwise much loss will take place, and the percentage of round coal will decrease, as pillars deteriorate when left standing long. With the greatest care possible, there is always some loss, varying from 7 per cent. to 12 per cent., in removing the pillars, and sometimes considerably more is lost through careless working and not setting in sufficient props.

The pillars themselves are extracted by taking slices, lifts, or judds off them, varying from 12 to 20 ft. in width,—15 ft. being a very

common lift. In this way they are reduced in size till a small pillar about 8 ft. square is left, this small pillar being then extracted as rapidly as possible. Fig. 114 shows one of the methods of reducing the pillars by taking a 'lift' off each side simultaneously.

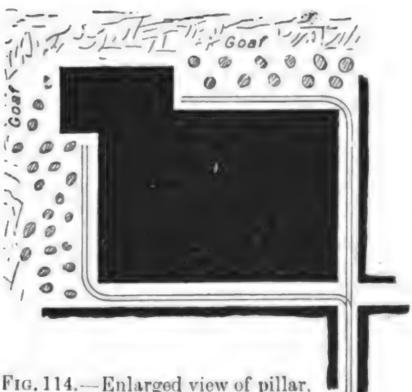


FIG. 114.—Enlarged view of pillar, showing method of extraction.

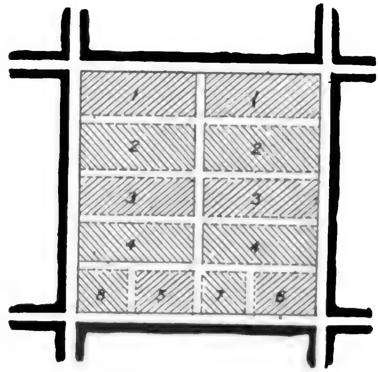


FIG. 115.—Extracting pillars.

If the pillars left are large and square, they are often split into two by driving a road through their centre, and then extracting the remainder by taking 'lifts' right and left as shown in fig. 115.

In this way the length of lift is shortened, which in the removal of pillars is a considerable advantage. Fig. 116 shows other methods of extracting the pillars which are sometimes adopted.

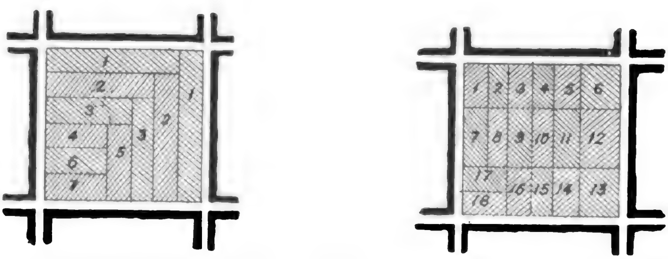


FIG. 116.—Extracting pillars.

In some districts, instead of the pillars being taken out in lifts, the whole pillar is extracted in one operation, and the waste left packed with rubbish as in ordinary longwall working. It is not, however, always convenient or even safe to attempt this method.

When the roof is bad, and fire-damp is given off freely, extracting the pillars becomes a still more dangerous operation. To be success-

ful in this part of the work, the following rules ought to be carefully attended to:—

No naked lights ought to be used in withdrawing timber, whether gas has been found or not, and the timber should be withdrawn when only a few men are in the pit.

The lifts should be made as short as possible and not too wide.

They should proceed regularly and as speedily as possible.

Two 'lifts' should not meet each other; one should be finished and the timber withdrawn before the other comes forward.

The timber ought to be withdrawn as soon as practical after a 'lift' is completed.

If plenty of timber be used (about one prop per square foot) less will be lost, and a plentiful supply of timber should be kept as near as possible to the working faces.

Panel System.—In the old method of working, the whole royalty to be worked was first cut into pillars right to the boundary line before any of the pillars were extracted; this entailed a large amount of loss, due to the length of time the stoops or pillars stood before being taken out, for where the roof was bad or the coal tender, they would be much crushed. To overcome this difficulty, workings are now often laid out in sections or panels, and as soon as one section is turned into pillars, these are at once extracted without being allowed to stand any length of time. This method is found both cheaper and to result in the production of better coal. At Hamilton Palace Colliery this system of working was adopted, the whole field being first divided into large blocks or panels, 336 × 356 yds., by pairs of headings or levels driven close to each other (fig. 117) and 'throughers' driven for ventilation. The levels and headings were driven by Stanley coal-heading machines, two machines working a level each with only a rib of coal 1 ft. thick left between them (fig. 118). This rib of coal obviated the use of brattice, and served to preserve the roof; the rib was subsequently taken out and the two drivages converted into one level 11 ft. wide, which was quite sufficient for a double road being laid down for haulage purposes. As the large blocks are formed, men are immediately set to work to form smaller pillars, 30 yds. by 20 yds., and as soon as these are driven a third set of men proceed to extract them. The great advantage of this method of working, as compared with that of forming pillars over large areas, is that they only stand for a short time after being formed, better coal being got, while a larger number of men can be employed, and hence a larger output is obtained.

At Clifton Hall Colliery, in the Manchester district, the Doe coal seam is worked on a similar principle. Pairs of levels are driven to the boundary, with 40 yds. between the openings, and 200 yds. above these another pair of levels is driven parallel. When the boundary is reached, a pair of headings is set away to connect them, thus dividing the coal into large blocks 200 yds. square. These blocks are then sub-divided into pillars 30 yds. square, and when a

block has been turned into pillars the latter are at once extracted, beginning at the boundary and working backwards. Lifts of 12 yds.

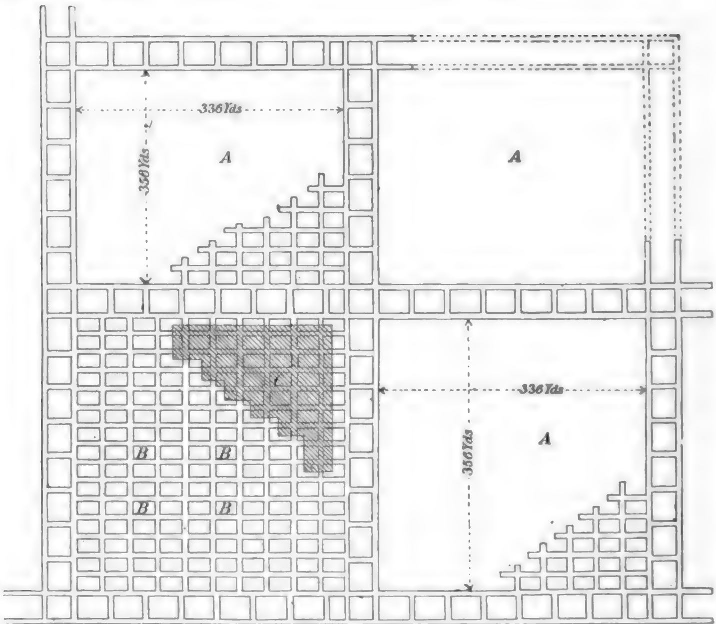


FIG. 117.—Panel system.

AA = Large pillars or panels. *BB* = Small pillars. *CC* = Pillars being extracted.

wide are taken off the pillars, a packing 9 ft. wide being carried up alongside the road—the material for which is inferior coal.

The seam is altogether 9 ft. thick, but the top $3\frac{1}{2}$ ft. is left on for a roof; the holing is done in the 7 in. of fireclay, which is holed for 3 ft. The top coal is then taken down and another 3 ft. holed, until 9 ft. of the bottom coal is bared—this part of bottom coal being obtained by blasting. The lifts off the pillars are always taken to the full rise of the seam.

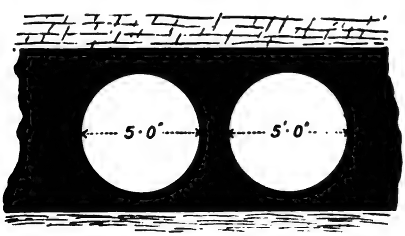


FIG. 118.—Machine levels or headings.

Special Methods of Working.—When two or more seams are close together, with little strata between them, it is often difficult to determine the method of working best adapted for getting all the coal out in good condition, particularly when the strata between the

coal seams are soft and friable, and when the seams themselves are largely intermixed with bands of dirt.

When two seams are separated by 6 or 10 fms. of strata, it is, as a general rule, best to work the lower seam out first, particularly if much water is given off, and to let the upper one stand for two or three years. By that time the roof will have settled down gradually and evenly. This plan may cost more than if the upper seam were worked first, but far better results are obtained as a rule. Of course, there are exceptions ; for instance, if the top seam had a bad roof, it might be better to work it first, or there may be a stipulation in the lease that the upper seam be first exhausted. When two seams are separated from one another by only a small thickness of rock, and if this stratum be hard and firm, it is better to work the top seam first, allow the roof to settle down, and then work the lower seam. If the intervening stratum be loose and friable, the bottom seam

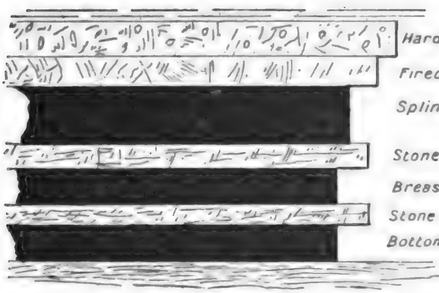


FIG. 119.—Section of seam.

should be worked first, as otherwise it may be found impossible to work it at all. Then, again, sometimes a number of seams occur together, separated only by thin bands of dirt or inferior coal. In a seam having a section, such as in fig. 119, the bottom portion is worked first up to the bottom of the Splint coal, leaving this on for a roof, while the stone and holings pack the waste, the working being carried on in the regular longwall way until the boundary is reached. Then the top part (Splint coal) is either wrought back from the boundary to the shaft or worked inwards in the usual way, the old roads serving for this working. The main object to be kept in view in working two seams close together is to get as much of both as possible, and not to damage either by working the other.

In Lanarkshire, the Splint and Virgin and the Kiltongue seams are often divided by bands of dirt varying from 1 ft. to 6 ft. in thickness, which renders them difficult to work, especially when the dirt between the two portions is 4 ft. to 6 ft. thick.

In the Blantyre district, the Splint and Virgin seams are separated

by a band of dirt only 1 ft. to 3 ft. thick, and in some of the collieries these seams are worked in two portions. The Virgin seam is worked in the ordinary longwall method, the walls being 14 or 16 yds. long. This coal consists of the Virgin seam and the ply of stone up to the bottom of the Splint coal; but to give sufficient height in the roads a portion of the latter is also ripped down. The walls are packed with the intervening stone, and with any stone resulting from falls of roof on the roads. The Virgin seam is worked in for a considerable distance in this manner, and then the Splint coal is worked back, what was the packing for the lower portion being now the holing for the top part. The one set of roads serves for both seams. By this method of working, round coal is economically got, but the main roads are extremely difficult to keep in repair, owing to the insufficient packing of the walls and the weight from the roof.

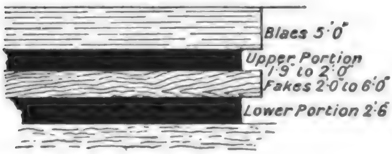


FIG. 120.—Section of Kiltongue seam.

Fig. 120 represents a section of the Kiltongue seam as it is found in the Coatbridge district. A common method of working it is as follows:—The bottom portion is worked from 9 to 12 ft. in advance of the upper portion, and short props *p p* are put up along the wall to support the intervening strata (fig. 121). When the coal has been taken out in the lower seam, the short props are then drawn, and

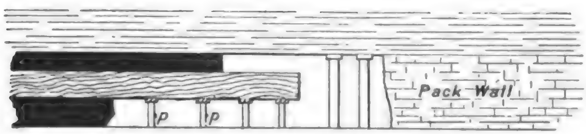


FIG. 121.—Method of working Kiltongue seam.

the fakes allowed to fall, a pack being put in along the face until all the loose material is used up. The top portion is then either blasted or wedged down in the usual way, care being taken to properly support the roof immediately the top coal is removed. This method works very well when the thickness of the strata between the two seams does not exceed 3 ft.; when it is greater it is dangerous to work by this system, owing to the difficulty of getting out the short props, and the danger arising from large slabs of the fakes breaking off and falling over at the face. Any open spaces along the face should be supported by wood pillars. Another method of working this is to keep both portions going as separate workings, and when the road is ripped to put in a short pack in both seams at the same time.

When the dirt separating the two seams is 5 or 6 ft. thick, the following method of working is often adopted :—The bottom seam is worked first by ordinary longwall. Main levels and headings are set away every 40 or 50 fms., and the drawing roads cut off on one side, and new branch roads are opened on the opposite side of the heading. As the branches are thus cut off in a section, the upper seam is opened up in the old road-heads and the seam worked back towards the main levels (fig. 122), beginning at the level itself, which would require to be worked during the night shift.

By cutting up on the 'low' side of the first branch road and the 'rise' side of the level, means are afforded for an air-current to circulate. In working by this method, the walls and roadways in the lower seam should be built very carefully, otherwise great

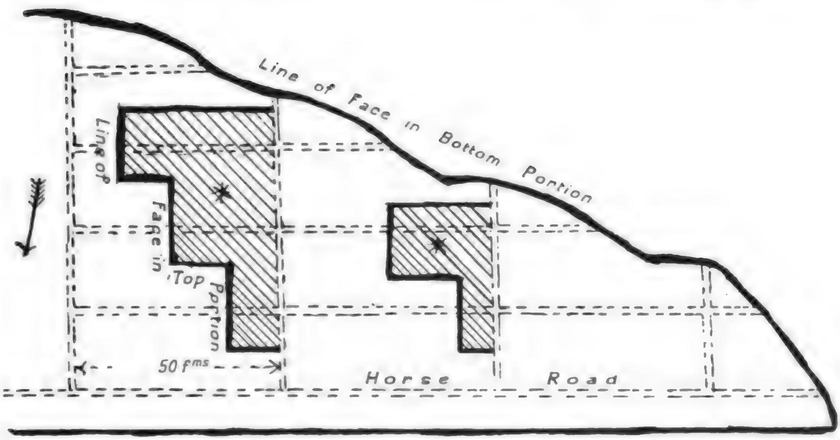


FIG. 122.—Working special seams.

difficulty is sometimes experienced in reaching the top part of the seam.

The advantages claimed for this method are:

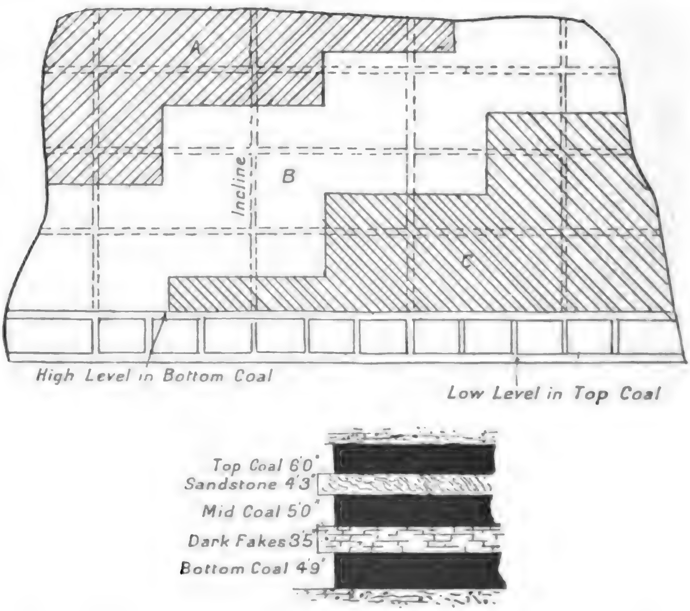
It is safe to work
Little ripping is required, and keeping roads open is dispensed with.
Less expense is incurred for timber.
A larger output is got by working both seams practically at the same time.

Sometimes three seams are found within a few feet of each other, as in some parts of the Ayrshire coal-field. This greatly increases the difficulty of working. The section here given (fig. 124) represents one that occurs at Dalmellington.*

The method of working adopted is very similar to that just described for two seams occurring close together.

* *Ibid.*, vol. xiv. pp. 113–114.

In this working two levels are driven—the low or main level in the top coal and the high level in the bottom coal. Sometimes, when the roof of the top portion is not very good, the main levels are driven in the bottom coal. Level cross-cuts are driven at convenient distances—usually 40 to 50 fms.—apart, from the top coal in the low level to the bottom coal in the high level. In this coal longwall workings are opened out, a barrier being left between the two levels, and branch roads 12 yds. apart are set off to the rise at right angles to the level (fig. 123). In the roads of



FIGS. 123, 124.—System of working adopted at Dalmellington in Ayrshire.

this working the dark fakes are ripped down to the bottom of the mid coal, the roads being packed 12 ft. wide. When the branch roads are worked up 40 or 50 fms. they are cut off by a new level; only those in a line with the cross-cuts are kept open as main roads, being afterwards fitted up as self-acting inclines to convey the coal to the low level.

When a certain area has been exhausted in the bottom coal, a new set of operations is begun at the cousie * level, by piercing up into the mid coal and opening out longwall workings in that seam, the roads of the lower seam being used. A pillar is left to protect the cross level, should this still be in use when it is reached. The roads

* Self-acting incline.

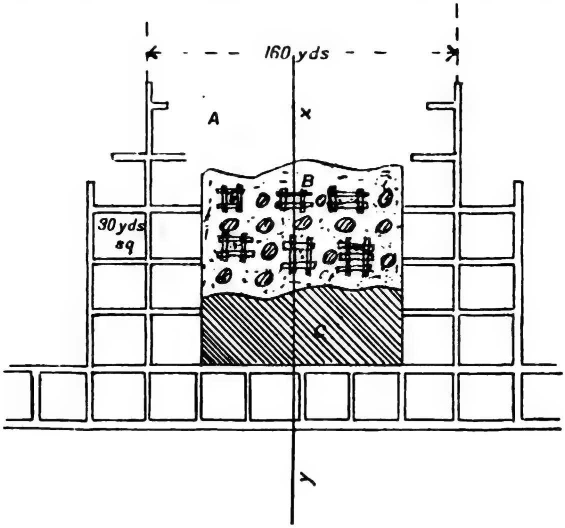
are finally ripped up to the bottom of the top coal, and a pack put in on the top of the dark fakes, which was the roof of the bottom coal, but is now the floor of the mid coal. When the mid coal has reached its limit, the top coal is pierced up along the faces and worked backwards longwall, there being no ripping in this working. Stones from the road-sides and old timber are used for pillars on either side of the road-head (gateway), the walls being propped in the usual way. By this method very little coal is lost, and the percentage of round coal got is larger than when the seams were worked by stoop and room.

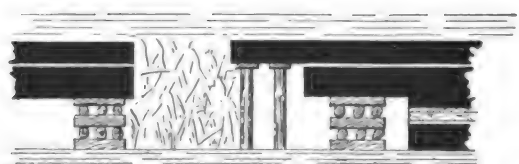
Spontaneous Combustion.—In many districts spontaneous combustion occurs when the coal is being worked. In the South Staffordshire thick coal underground fires are of frequent occurrence, and in the Dysart thick seam in Fifeshire the coal very readily takes fire on being worked. These underground fires are more or less due to the presence of quantities of dross or inferior coal in the waste. The theory, or rather theories—for there are many—of spontaneous combustion in underground workings need not be discussed here, and only the methods for dealing with such occurrences will be dealt with. To overcome the danger arising from underground fires, various means have been adopted. Where the seam is worked on the longwall principle, continuous walls of clay have been employed to prevent the air from entering the waste; banks of sand have also been used for the same purpose, but neither method seems to be very practicable. When a 'crush' came on, a clay wall would, in all probability, be pushed out into the roadway, while such walls would certainly entail a large amount of extra labour and expense in construction.

Where these underground fires are of frequent occurrence, it would seem best to work the seam on the 'retreating' system, *i.e.* driving the principal roads out to the boundary, and keeping them dipping from the shaft if possible; the coal could then be worked uphill towards the shaft, and the waste allowed to fill with water up to within a convenient distance from the face.

To overcome the danger from underground fires in Fifeshire, the coal is sometimes worked on the panel system. The coal is thick, and is generally worked in two portions (figs. 125, 126, 127). During the first working extraction extends to the 'cherry' coal, the levels being driven 8 ft. wide, and pillars 30 yds. square left on the 'rise' side. The cross 'throughers' are also driven 8 ft. wide to allow double roads to be laid, as the latter are worked as self-acting inclines. The panels are generally formed 140 to 160 yds. square; and in forming the panel a main heading is driven 8 ft. wide, and pillars formed on each side 30 yds. square. The heading is driven up 80 yds., thus forming two pillars and part of a third one. The regular working is now begun. The stalls are driven 15 ft. wide, and small coal pillars, 6 to 8 ft. square, left to support the roof.

No dross is filled in this working. Wood pillars are put in, generally three in each place, this being sufficient to allow the roof to subside gradually. This method is continued until the 30-yd. level above is reached; it is then cut off and the coal brought out on the upper road. While working thus, the abandoned workings left behind will subside from 5 ft. at the commencement to a height of 2 ft. 6 in. A margin of 10 ft. is left next the upper level for protection, and the timber is taken out and the dross filled, the





FIGS. 125, 126.—Dysart thick seam working.

small 8-ft. pillars being now crushed to dust. The coal is then worked back, and every place of ingress built up, to prevent air travelling through the abandoned workings. When the boundary has been reached, the large pillars forming the barriers are worked back.

The advantages claimed for this system are that practically all the coal is extracted, and in the event of any fire occurring it can be easily shut off by driving a level higher up and opening out anew. By this method of working, however, a large proportion of the

coal, and particularly of the pillars, is so badly crushed as to be of relatively little commercial value. The seams at most of the collieries in Fife are now worked on the two-fold system.

Working Highly-inclined Seams.—In working seams that are

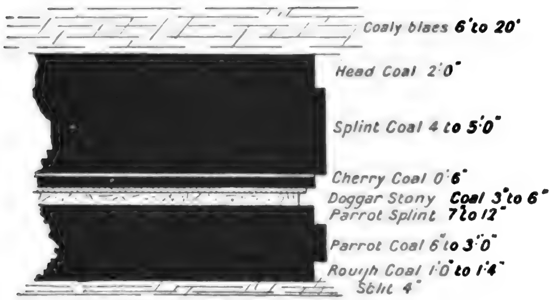
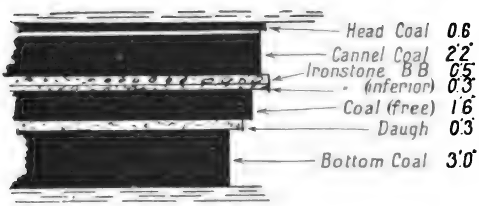
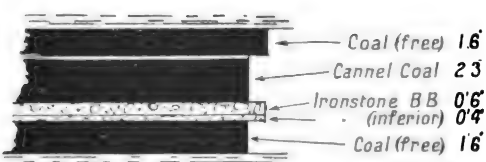


FIG. 127.—Section of Dysart thick seam.

highly inclined, from 70° to 90° from the horizontal, it is found, as a general rule, that whether the seams are moderately thick or very thin, the best system of working is by longwall or some modification of it; especially is this the case when the seams reach a certain depth, say 120 to 150 fms. from the surface, when the pressure in



Section of Great Seam



Section of Stairhead Seam

FIGS. 128, 129.—Sections at Niddrie.

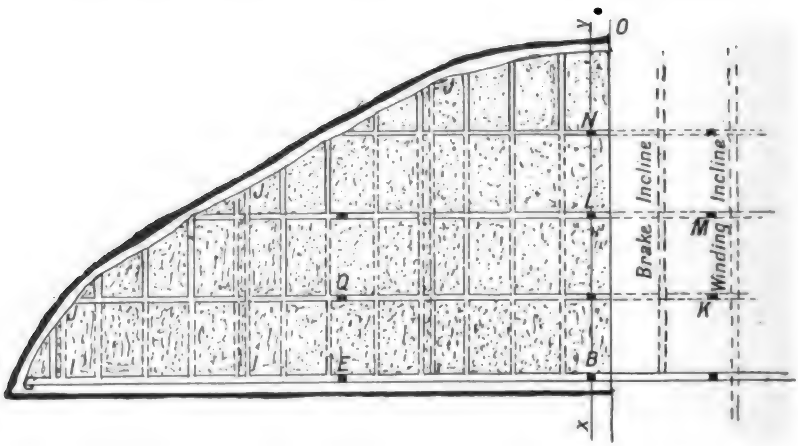
highly-inclined seams becomes very great. Until they reach such depths, they may, however, be worked perfectly well by 'bord and pillar' in the ordinary way.

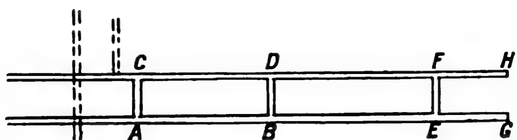
At Niddrie Collieries, near Edinburgh, where the seams are inclined at an angle of 65° to 90° from the horizontal, longwall is now the

only system that is practised, although formerly the seams were worked by 'stoop and room.'* Figs. 128, 129 give a section of the two seams most largely worked. The coal is won by both inclined and vertical shafts, and is worked in lifts of from 60 to 80 fms. in depth, divided into panels of about 200 fms. in length.

In each panel, and preferably as near to its centre as possible, a 'brake' incline is formed, by means of which the coal is lowered from any intermediate roads to the bottom level, along which it is conveyed to the winding incline or shaft.

When the dip does not exceed 70°, the method of working is as





FIGS. 130, 131.—Longwall system at Niddrie

follows:—Close levels (A, B, C, D, figs. 130, 131) are driven in both seams from the winding incline at the depth fixed upon as the bottom of the lift, and when a sufficient distance has been reached they are connected by a cross-cut (B D, figs. 130, 131), and longwall operations are then commenced. A level about 8 yds. wide (B, E, G, figs. 130, 131) is set away, leaving 6 ft. of stowage under the rails; the rise side of the place being continuously timbered with pillars 3 ft. thick and built alternately draught-board fashion, the open spaces being filled with dross. 'Spouts' or 'shoots,' 3 to 4 ft. wide, built and causewayed with pieces of ironstone, are branched

* Mr Hugh Johnstone, *Trans. Min. Inst. Scot.*, vol. x. p. 204.

off straight to the rise 12 to 16 yds. apart from centre to centre. The goaf is stowed with the daugh or fireclay, dross, rough coal, and any ironstone not required for packings. For convenience in working, the walls are so arranged that each has a long 'rise' side and a very short 'dip' side. The 'cannel' coal, which is the part for which the seam is principally worked, is dropped down the spouts (I J, fig. 130), at the bottom of which it is filled by a 'drawer.' At intervals of about 70 yds. travelling roads (L B, fig. 130) are formed to afford convenient access to the working places at different points. These are built similar to the spouts, and are furnished with ladders.

While the longwall is progressing, roads (K, L, M, N, fig. 130) are driven in the Stairhead seam at intervals of 40 yds., and cross-cut mines (G N, fig. 130) are driven from thence to the Great seam, so as to strike the latter before the longwall headings reach their level. From these mines, intermediate levels (L N, fig. 130) are carried across the working faces as they come up, cutting off the 'spouts,' or what would be, in ordinary longwall, branch roads. The rails for these intermediate levels are laid upon the stowage, and the rise side of the road is timbered similarly to the levels. The close level in the Stairhead seam (D H, fig. 131) is carried in advance of the level in the Great seam, and from it cross-cuts (F E, fig. 131) are driven, connecting the two seams at intervals of 60 to 80 fms., for the purpose of cutting off the out-bye portion of the Great seam level as soon as the 'spouts' on it have been cut off by the intermediate level above. The same system is followed with the upper levels, the object being to 'shorten the life' of the roads in the Great seam and to keep the horse or engine haulage as near to the working faces as possible. The method of building the levels and spouts is shown in figs. 132, 133, 134. The stowage on the dip side of the levels tends to prevent the roof breaking and bursting out on the roads. The roof is supported by 'half-rounds' (D, fig. 133), 8 in. × 4 in., placed 4 ft. apart and supported at each end by props 5 in. diameter.

The walls are either 12 or 16 yds. long; in the first case two men usually work in each, and three men when they are 16 yds. in length. Very few props are used at the face, but the coal is kept closely 'spragged,' the distance between the 'sprags' or 'holing props' being not more than 3 or 4 ft. When the seam lies at an angle of 90° or in a vertical position, a rather different method is adopted. The plan shown in fig. 135 will explain the system.

The brake incline and haulage roads are made in the Stairhead seam as before, and close levels are branched off this incline at distances 18 yds. apart, and from each level a cross-cut mine is driven to the Great seam.

In the bottom level (A B, fig. 135), 6 ft. of stowage is kept below the rails, and the rise side of road is well timbered. The height of the roads is 5½ ft. clear. The 'rise' side of the place is kept

trailing so as to form an angle of 45° with the road. As soon as this level has been opened up sufficiently to let the rise side reach the level of the cross-cut mine (C), roads from this mine are laid on the top of the stowage, their rise sides being timbered, and the working

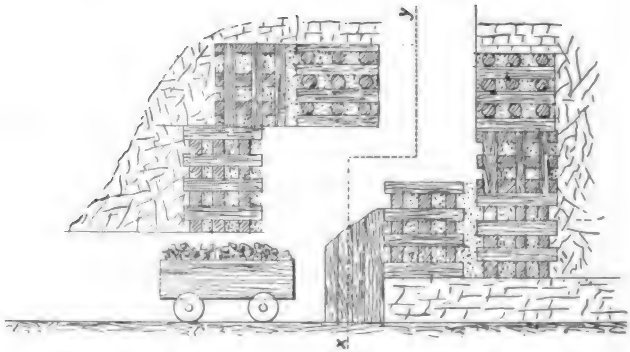


FIG. 132.—Method of supporting 'shoots.'

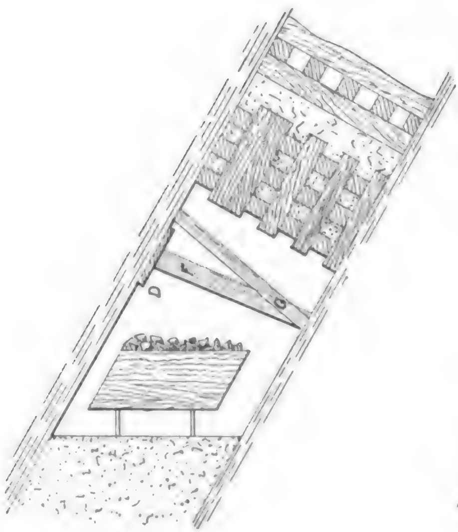


FIG. 133.—Method of supporting level.

FIG. 134.—Face of working.

is again extended to the cross-cut above (D), and so on to the top of the brake incline.

The roof in this working forms one side of the road, and is supported by half-round crowns, 8 in. × 8 in. × 4 in. and 4 ft. apart,

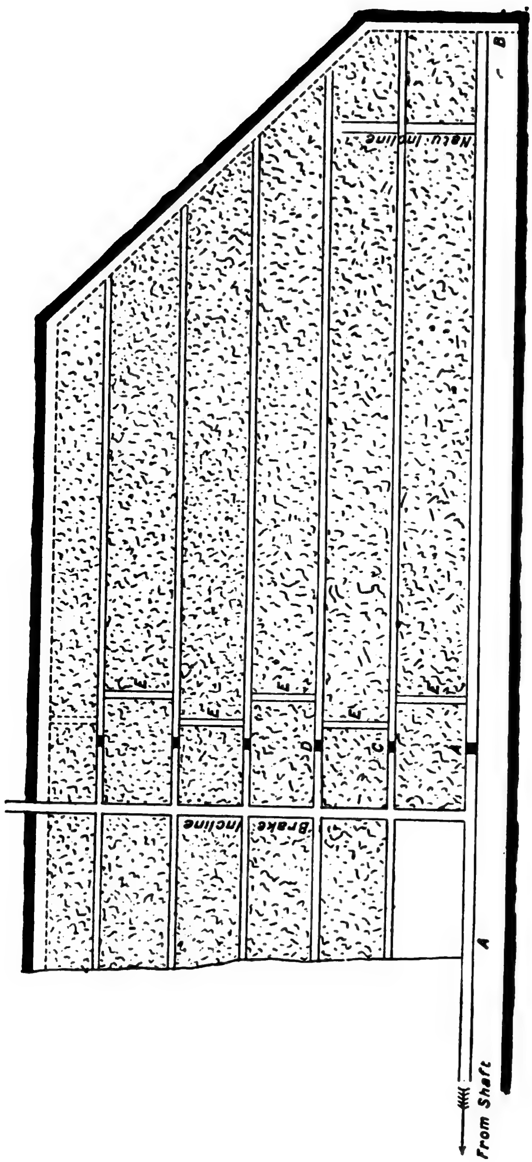


FIG. 135.—Plan of working at Niddrie when the seam is vertical.

the upper end of the crown being built into the timbering and the lower end into the stowage. The brake inclines are usually 200 fms. apart, and the coal is worked for a distance of 100 fms. on each side.

At Kilsyth, Stirlingshire, the seams sometimes lie at an angle varying from 30° to 45°, and here both stoop and room and longwall working are practised. A main incline is driven right from the surface to the dip of the seam, and the coal is worked in 'lifts' or 'benches' of about 100 fms. each. At each of these 'lifts' levels are set away on each side of the main incline (A B, fig. 136). These levels are filled on the dip side to make the rails lie as horizontally as possible. From these levels, which are usually 12 to 15 ft. wide,

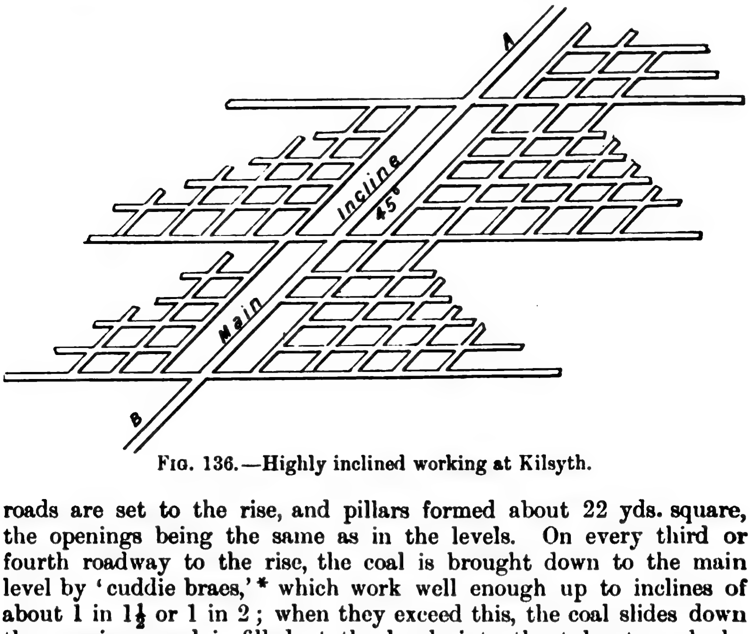


FIG. 136.—Highly inclined working at Kilsyth.

roads are set to the rise, and pillars formed about 22 yds. square, the openings being the same as in the levels. On every third or fourth roadway to the rise, the coal is brought down to the main level by 'cuddie braes,'* which work well enough up to inclines of about 1 in $1\frac{1}{2}$ or 1 in 2; when they exceed this, the coal slides down the openings, and is filled at the levels into the tubs, two planks being put across the mouth of the opening to prevent the coal from sliding out into the main levels.

As the coal is chiefly used for coking, the percentage of dross is of no consequence, pulverisation being, in any case, necessary. After a lift of 100 fms. has been turned into pillars, these are themselves extracted, by taking as many 'slices' as possible across level course, and a few lifts to the full rise. The tubs are drawn up the main incline in sets of six or seven, each tub holding about 8 cwts.

* Term used in Scotland for balance incline or jig brow.

of coal, and being filled only level with the sides. A comparatively wide gauge roadway is used, the width between the rails being 3 ft. 2 in. This is found necessary to prevent the tubs from over-balancing in the levels.

Main inclines from the surface work very well when the distance is not very great (300 fms. or so), and when the inclination is steep enough to admit of a cage being used, but where the inclination is between 30° and 45° vertical shafts are to be preferred, as offering better facilities for winding and for dealing with water.

At the Shamrock Colliery, Westphalia, where the seams dip at an

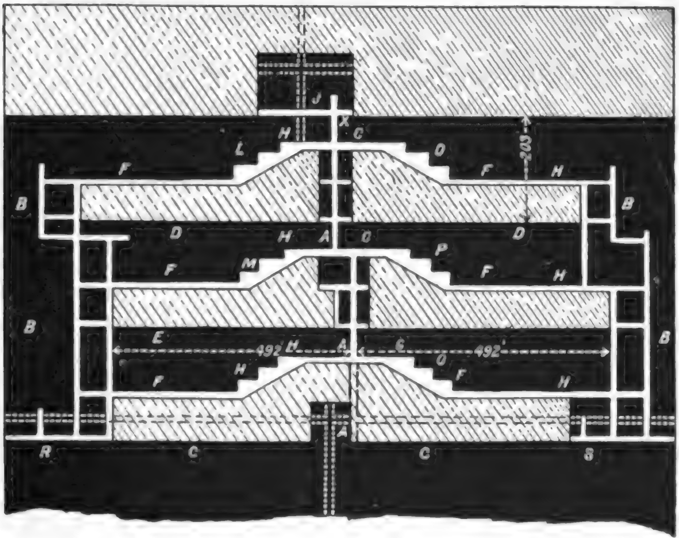


FIG. 137.—Method of working at Shamrock Colliery.

angle of 45°, a rather peculiar and complicated method of working is carried out.* The seam worked is from 7 to 8 ft. thick, and consists of a single bed of coal without any bands of dirt, the holing being done next the floor. The method of working adopted is illustrated in fig. 137. It is carried out in stages or panels of 200 yds. from top to bottom, each panel being served by a main or bottom level. The ground between two main levels is divided into great blocks or pillars 330 yds. wide, by headings B B driven to the rise, which serve as self-acting inclined planes for letting the coal down to the bottom level. These large blocks are subdivided horizontally into three parts by two intermediate levels, and into two parts on the

* *Lectures on Mining*, subj. 5, pp. 32-35, by Prof. Wm. Galloway.

line of dip by the central heading A, which is used for sending rubbish down into the workings.

At the point where the central heading A intersects the higher level, a small shaft X, 20 ft. deep, is sunk to serve as storage room for this rubbish, which is used for filling up the workings. The bottom of this shaft is connected to the central heading A by a short, level, cross-measure drift. All the rubbish which results from the driving of cross-measure drifts, and from the enlargement of roadways, etc., as well as that brought from the surface, is tipped into this shaft, from which the men who attend to the stowing of the workings derive their supplies, which are subsequently sent down the central incline A into the workings of the three sub-stages. The three sub-stages provide six level-course working places, three at the points L, M, N, and three at the corresponding points O, P, Q, on the opposite side of the heading A. While the three working places on one side of the central incline are producing coal, the three on the other side are being stowed, and *vice versâ*. When the work of stowing the three places on one side has been completed, the stowers and miners exchange places. Protection pillars having been left on the sides of the central incline, these are now taken out and replaced by stowing after the sub-stages have been exhausted. The lower end of the central heading is always stowed up carefully at the commencement, in order that the air which enters by the lower cross-measure drift may be compelled to pass to the right and left of the sub-stage workings. It then finds its way up the coal inclines, along the haulage roads to the working places, along the faces to the central incline, and through the latter into the next higher level.

In the accompanying illustration, A A are central inclined planes, in which the rubbish is let down from the higher level; B B are inclined planes for letting down coal into the lower level; C C, bottom level of the stage which is being worked, stowed with rubbish; D D and E E, levels of the sub-stages, also stowed up; and F F, roads along which the coal is hauled from the working places to the inclines.

G G are roads along which the stowing is hauled from the central incline to the working places. After these have been driven out to the limit of the sub-stage, these roads become hauling roads for coal from the working places next above. J J are protection pillars, and H H doors provided with regulators for distributing the ventilating current to each set of working places. L, M, and N are working places which, after producing coal for a certain time, are eventually stowed up, and O, P, Q are places which are in process of being stowed with rubbish, and are afterwards occupied as working places for coal.

R S are short cross-measure drifts to a lower seam, 16 in. thick, in which a level is driven under the workings of the lowest sub-stage to form a communication with the main cross-measure drift for haulage and ventilation. The cost of working by this method, with complete stowing, is returned at 2/0·7d. per ton got, including the

cost of landing the coal at the bottom level. It was found from experience that the additional proportion of large coal obtained by this system of working compensated to a great extent for the additional cost of complete stowing.

At the Tesla Mines, Alameda County, California, seams of coal are worked which have an inclination of 60°.* In one of these seams they have adopted a system called the Angle Method of Working. This system is illustrated in fig. 138.

In this system the gangway chutes are driven at right angles to the strike of the seam 40 ft. up the pitch; a cross-cut 6 ft. by 5 ft. is then driven parallel with the gangway. From this cross-cut, chutes are driven at an angle of 35°, at the same distance apart as

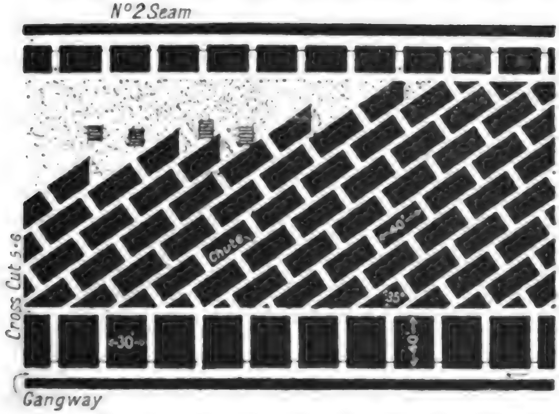


FIG. 138.—Angle system of working at Tesla Mines, California.

the gangway chutes (30 ft.); cross-cuts for ventilation being driven every 40 ft. between the chutes. After a panel of five or more chutes has been driven for the required distance, work is commenced on the upper outside pillar, the pillars on that line are drawn and the next line of pillars attacked, and this is continued until the panel or block is worked down to the cross-cut over the gangway. About every 80 ft. in this level it is found advantageous to pack a row of cogs, parallel with the strike of the seam, as the pillars are taken out. This serves to prevent the crushing of the pillars, and prevents any accidents from falls of rock.

In the regular working places little timber is required, as the roof is very good. It is claimed for this system of working that the coal undergoes less breakage than in the method where vertical chutes are used, while in the matter of timber a considerable saving can be effected.

* *Mines and Minerals*, Oct. 1898.

In the No. 7 or Summit seam a rather different method has been adopted. This seam averages 7 ft. of clear coal without foreign matter of any kind. The roof is good, being composed of a shelly slate, while the floor is a hard, compact sandstone, and it is in this that the chutes, airways, inclines, and other openings necessitated by this method of working and the characteristics peculiar to the seam are driven. The method of working this seam is shown in fig. 139. Two double chutes are driven up the pitch, or to the full rise, at a distance of 36 ft. apart, connected every 40 ft. by cross-cuts, one side of each chute being used for coal and the other side as a manway and air course.

When the seams are thin and highly inclined, a method of working termed '*Gradins Renversés*' or 'inverted steps' is often adopted in

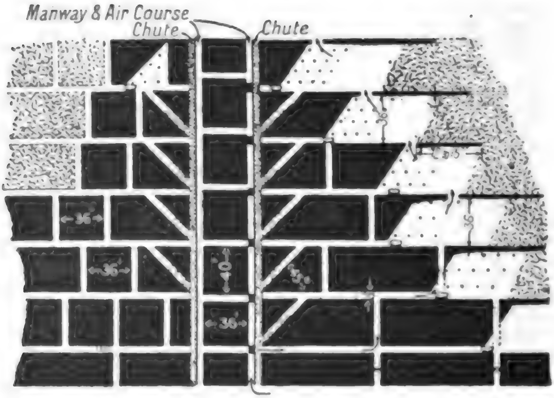


FIG. 139.—Method of working the Summit seam at Tesla Mines, California.

Belgium and other European countries. The seam is worked in the longwall method, with the face advancing in the line of strike. The system is much like that adopted at the Niddrie Collieries, which has already been fully described. The working face advances in the direction indicated by the arrow (fig. 140). The miner, standing on a planking beneath him or perched on the props stanchioned between the roof and floor, operates on the vertical face of the coal in his stall, while solid coal overhangs him. D is the drawing road leading to the main haulage road, and receives the coal from the stalls through the chimney C C, left in the gob or waste. Each chimney is provided at the bottom with a hopper door, which is opened when a tub is brought under it to be filled. As far as possible it is sought to keep the chimneys full of coal, but, even with careful supervision, this is not always attained. An obvious inconvenience attaching to these chimneys is the liability of the coal to get jammed, while they always present a source of danger to the passage of men

over them. The height of a step or stall, which varies in different localities from 2 to 4 yds., is determined by local circumstances, chief among these being the nature of roof and pavement, the amount of gas constantly given off from the coal, and the liability to sudden

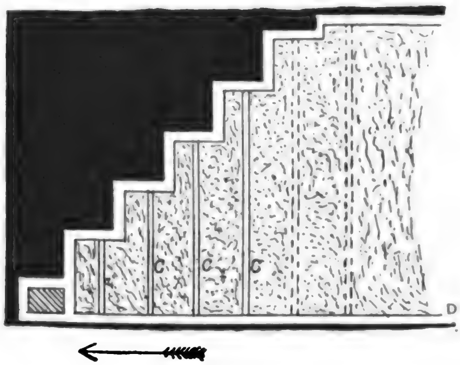


FIG. 140.—*Gradins Renversés.*

outbursts of gas. Other things equal, the height of each step regulates the rate of daily advance.

When the height given to a stage is considerable, one or more intermediate or false levels (A A, fig. 141), communicating with the

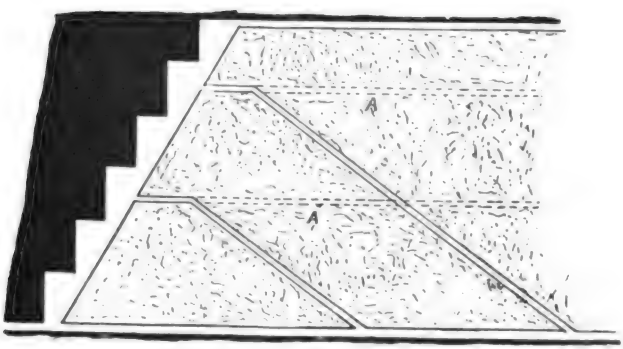


FIG. 141.—Working by 'inverted steps.'

lower level by means of inclined planes through the gob, serves to divide the distance and to limit the length of the chimneys.

Double and Single Stall Methods.—In South Wales a method of working is adopted, which is peculiar to that district, although, in some of the other coal-fields of Britain, modifications of this system of working are sometimes met with.

This is the single stall system, in which, instead of a pair of narrow levels being driven as in ordinary pillar and stall working, a stall about 8 yds. wide is carried forward.

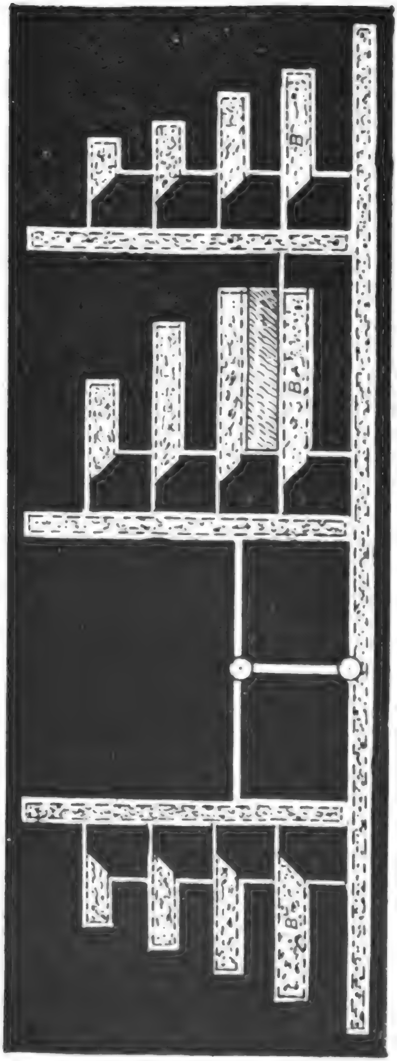


FIG. 142.—Single stall system.

Any rubbish that is found in the seam is tightly stowed in the place as it advances, a drawing road being formed on the down-side next the coal, while on the up-side an air-way is formed. When the main levels have been driven out a considerable distance to form a shaft pillar, headings of the same width as the levels (8 or 9 yds.) are set away to the full rise, with the drawing road and air-way formed in the same way. From these headings, stalls, parallel to the main levels or level-course, are branched off every 20 to 24 yds. between centres. These stalls are driven 6 or 7 ft. wide for a few yards, when they are opened out to 10 or 12 yds., according to the amount of rubbish available and the depth from the surface. The stalls are worked in exactly the same manner as the levels and headings, the roadway being formed on the down-side of the road and the air-way on the rise, the space between firmly packed.

Headings are driven off the levels at distances of 100 yds., forming a sort of panel, and a narrow drift is formed in each of these panels between the main level and the lower stall (B, fig. 142) to admit of air passing from the main level into the stall workings.

The stalls of the heading are driven out for a distance of 75 yds., and then they are driven narrow for a distance of 15 yds. or so, until the next heading is cut into, thus forming a connection between the different headings for ventilation. When the second heading has been put up and stalls formed off, parallel, and opposite to those on the first heading, the stalls first driven are then worked (to the first heading), leaving a pillar 15 yds. thick to protect the headings. The pillar between the stalls is thus brought back to the required distance. The same process is carried on throughout, headings being turned off the main levels at regular intervals, and a block of coal is left alongside the main road until the latter has reached its destination, when it may be worked back towards the shaft.

The double stall system is similar to the single stall system, with the exception of the stalls being driven very much wider, and two roadways being formed instead of one roadway and an air-way. Main levels are driven out from each side of the shaft as before, but in

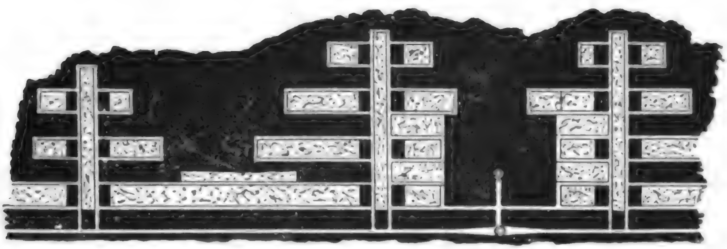


FIG. 113.—Double stall system.

double-stall working these levels are often driven narrow, 9 ft. or so, and headings are put away to the full rise, 8 yds. wide, a block or panel of coal of 90 yds. being left between the headings. From these headings roads are turned, parallel to the main level, but instead of these being single roads, as in the single stall system, two roads are set away with 18 yds. of coal between them, and after being driven a short distance they are opened out and connected, thus forming a place 22 to 24 yds. wide. The roads are formed one on each side next the solid coal, and the space packed between them, and in this way they are carried forward until within 15 yds. of the next heading, when one of the roads is continued through to the heading and forms an air-way. The pillar left between two stalls is then worked back, each road taking a slice equal to half the pillar or about 6 yds. This back-lift is brought to within 30 ft. or so of the heading, being left for protection until the latter is finished, when it may be extracted.

These systems of working are best suited for seams having a large amount of rubbish with which to stow the space between the roads properly, because if the separating pack is not well built the ventila-

tion will suffer. A large percentage of the coal in both systems is either lost or rendered useless through the small pillars or stumps left, for protecting the headings and levels, becoming crushed to dust, and it is a question if longwall would not be in any case better suited in districts where single or double stall is now worked.

Working Thick Seams.—In Britain, and also in other European countries, seams of great thickness are found, which are often difficult to work.

At Quarrelton, near Johnstone, Scotland, a very thick seam of coal is found, 60 to 90 ft. thick, in a trough or basin; but here the seam, being found near the surface, was worked chiefly on the open-cast principle. Shafts were also sunk to win some of the deeper and more thickly-covered parts, but the coal being of poor quality and ready to take fire, little has been worked.

In South Staffordshire the valuable 'Dudley Thick' or 'Ten Yard Seam' is found, of a thickness varying from 25 to 37 ft.

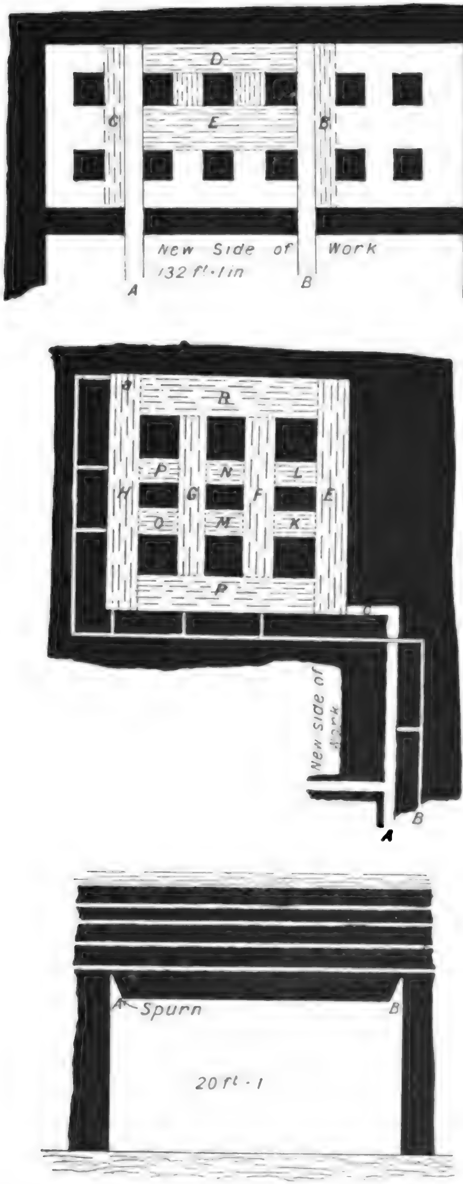
This seam has been worked at various times in three different ways, viz.:—(1) Square work; (2) longwall, the face being extracted in two portions; (3) longwall, the whole of the face being removed at one operation.

The first method is the one that is most largely and successfully followed, the other two methods being almost, if not quite, abandoned.

In working this thick seam, no matter what system is adopted, it is an invariable rule to first drive out the roads to the boundary of the royalty and then work back towards the shaft.

Square Work.—In this method of working after the gate-road and air-head (A and B, fig. 144) have reached the boundary of the royalty, the first operation is to drive a narrow head called a bolt-hole (C, fig. 145). When this has reached a distance of between 10 and 15 yds., its width is increased to 6 or even 9 yds., when it practically becomes a stall (D, fig. 144). In driving these stalls, the next step taken is to 'hole' or 'kirve' the benches for a distance of about 9 yds., the roof being supported by small cogs of slack and timber or by short sprags. The 'slipper' and 'sawyer' parts of the seam will then be vertically cut on both sides of the stall, 'spurns,' or intervals of solid coal, being left every 6 ft., as in the case of gate-roads. When these 'spurns' are removed, it frequently happens that the 'patchells' and 'stone' coal fall with the 'slipper' and 'sawyer'; if this occurs, the sides are dressed straight and timbering put in to support the roof. Further lengths of 9 yds. are similarly attacked in the stall (C, fig. 144), until the back rib is reached. While this has been going on, other stalls (E, F, G, fig. 145) have been driven in the directions shown by the dotted lines, and finally the pillars will be 'thirled' by means of the short lengths shown (K, L, M, N, O, fig. 145). After the driving of the stall R, the side of work is fully opened, and cutting in of the top coal begins.

This proceeds in the same manner, each layer or bed of coal being



Figs. 144, 145, 146.—Working thick seam in South Staffordshire.

removed in ascending order. If the roof is very good a long range can be taken, but it is far better to carry out this part of the work in short ones; distances of 20 yds. being taken and a timber cog built up to the roof. A vertical groove will then be cut on each side of the stall and on the face of the cog, treating the latter as if it were solid coal, spurns being left as before; their appearance when the 'brazils' and 'heath' coal are being cut is shown in fig. 146. When these 'spurns' are taken out (as shown at B, fig. 146) and the timber removed, the whole mass within the portion cut falls down and is ready for loading up. Similarly a layer of top coal is removed all over the side of the work, and when this is done the 'white' coal and then the top 'slipper' coal will be attacked, and the same cycle of operations gone through, until these seams have been cut in. No attempt is made to get the 'roofs'; sometimes, however, they break down without the superincumbent strata following, in which case they are removed as speedily as possible. In cutting the top coal, the workmen stand on a platform, constructed by cutting two holes in the solid coal and inserting short timbers, on which a plank is placed. A frequent occurrence is to build timber cogs up to the roof when the bottom coals have been removed; these cogs being placed in the openings between the solid pillars of coal, and when this is done the latter are reduced in a marked degree.

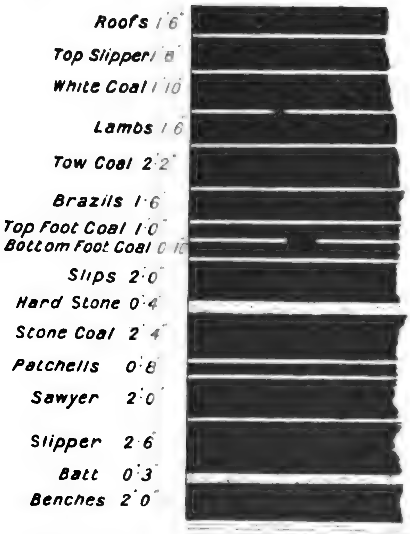


FIG. 147.—Section of 'ten yard seam,' Dudley.

The other two methods of working have not given such good results as 'square' work, owing to the large amount of dross produced, while a smaller yield per acre was likewise obtained by both the alternative longwall systems.

CHAPTER IX.

TIMBERING ROADWAYS, Etc.

Necessity for Timbering.—When roadways are opened out underground, they must be efficiently supported in order to keep them secure and safe for traffic. Upon the artificial supports which are generally used depend the lives of those employed, and the regularity of the coal output. The proper timbering of underground workings is therefore of the utmost importance. When it is remembered that about 50 per cent. of the total number of accidents which occur underground are due to falls of roof and sides in the roadways and at the coal-face, the necessity of paying considerable attention to proper spragging and propping becomes even more manifest.

A bad roof is often defined as one which requires the application of systematic support, while a good one is supposed to require no such methodical treatment. The principle which should, however, be observed is "that timbering is not for a bad roof only: it is intended to prevent a so-called good roof from becoming bad."

A roof may be bad in itself, owing to the slight cohesion of its particles, as in a fireclay roof. It is then called 'tender.' What may be a good, strong roof in itself may, on the other hand, be so full of 'lipes' and 'faults' as to form a very treacherous roof. As already shown (Chap. VIII., p. 137), the difficulty or otherwise of maintaining a roof will depend a great deal on the direction in which the roadways are driven, *i.e.* at right angles to the cleavage in the roof or parallel with it.

Mr A. R. Sawyer * says:—"It is fallacious to suppose that what are called bad roofs need be productive of more accidents than good roofs. Roofs containing numerous dislocations, lying much in the same direction, necessitate a specially systematic kind of timbering, which will ensure comparative safety; while a good roof, in which dislocations occur rarely, but unexpectedly, may be fraught with

Note.—The greater part of this chapter is from a paper on "Timbering and Supporting Underground Workings," read by the author before the Mining Institute of Scotland, December 1898, and published in the *Transactions of the Federated Institution of Mining Engineers*, vol. xvi. pp. 230–248.

* *Accidents in Mines*, 1886, by Mr A. R. Sawyer, p. 28.

danger from the want of a rigid method of timbering, irrespective of whether, in individual places, posts are absolutely required or not."

The timbering of the working is done sometimes by the miners themselves, and sometimes by special workmen. In Northumberland and Durham the whole of the timbering at the faces or walls is done by 'deputies,' these men devoting their whole time to the work, while the colliers at the face have nothing to do with the propping. In other districts, as in Lancashire, Wales, and Scotland, the whole of the timbering at the working faces, and often for some distance back in the roadways, is performed by the colliers themselves.

Much difference of opinion exists as to which is the better and safer method. The writer is of opinion that the system of making the miner responsible for the timbering is the one most likely to secure attention to the safety of the roof and sides. The miner, being continually employed in his own stall, is more likely to detect any change in the condition of the roof than a deputy who only visits the stalls at intervals. The miner, too, is always at hand to set a prop in cases of emergency, and will take an interest in keeping his place secure for his own sake. It has been urged that no man is more careless of his own safety than the miner, and perhaps there is some truth in this, even good, reliable men often falling victims to their own carelessness. While the miner ought to have the proper timbering of his working place in his own hands, more power ought to be invested in those supervising his work, such as firemen, to compel the adoption of a systematic method of propping and to put up props at stated distances apart, whether the roof appears good or not.

The Royal Commission, in their Report on Accidents in Mines, recommended attention to the following points:—

> The maintenance of ample supplies of timber in localities convenient to the workmen.
>
> The proper training of each miner to the best method of timbering and otherwise protecting his working place.
>
> The exercise of increased care on the part of the workmen in watching the roof, sides, and faces, and protecting themselves in time.
>
> The introduction, as far as possible, of arrangements with the workmen, which will make it their interest not to avoid the labour of putting up the necessary timber for their protection.
>
> The employment of special timber men or deputies for the timbering of the main-ways, and also for the repair as well as the drawing of timber.
>
> Preventing timber being left in the goaf of longwall workings which would have the effect of breaking the roof.
>
> Driving the working places as rapidly as possible, by shifts of an ample number of workmen at each face, and so reducing the risk of falls and exposing the least number of men to danger at any one time.

These instructions, if they could be rigidly carried out, would tend in a large measure to reduce the number of accidents to a minimum. It is, however, impossible to lay down any general rule for the timbering of mines which would be satisfactory under all circumstances, and applicable to all mining districts.

Timbering may be divided into two distinct operations, namely :—

Timbering the working faces.
Timbering or supporting the drawing and main roads.

The timbering or supporting of the roof may be done in a variety of ways, according to its nature and that of the floor and sides, and may be carried out as follows :—

By packing the waste entirely, where sufficient rubbish can be got, and timbering at the face and roads, as in longwall.

By partial packing of the waste or goaf, by buildings or stone pillars with intervening spaces, and timbering at the face and roads.

By timbering at the face and in the roads, and leaving wood or stone pillars in the waste as supports. This is the usual practice where the seam yields no packing material.

By timbering alone, without any packing whatever, as in bord and pillar, or in longwall on the retreating system.

Supporting the main roads with brick arching, steel or iron, or with combinations of masonry and iron, or steel work.

Timber.—For underground work the different varieties of timber used in Great Britain are larch, Scotch fir, Norwegian pine, and sometimes oak and beech.

For ordinary timbering Scotch fir is largely used, particularly where the timber does not require to be heavy, and where the pressure is not great; but in haulage and main roads that require to be kept open for some time, larch has been found to give the best results, both as regards durability and economy.

Oak is an excellent wood where great pressure is likely to be met with, but the cost of such timber is high, and it is difficult to get straight pieces of any length, while it is also a difficult wood to dress.

Beech props are short-grained, more or less brittle, and break off short under crushing stresses. Their great weight, too, as compared with other wood, and the consequent difficulty in handling them, militates against their use underground.

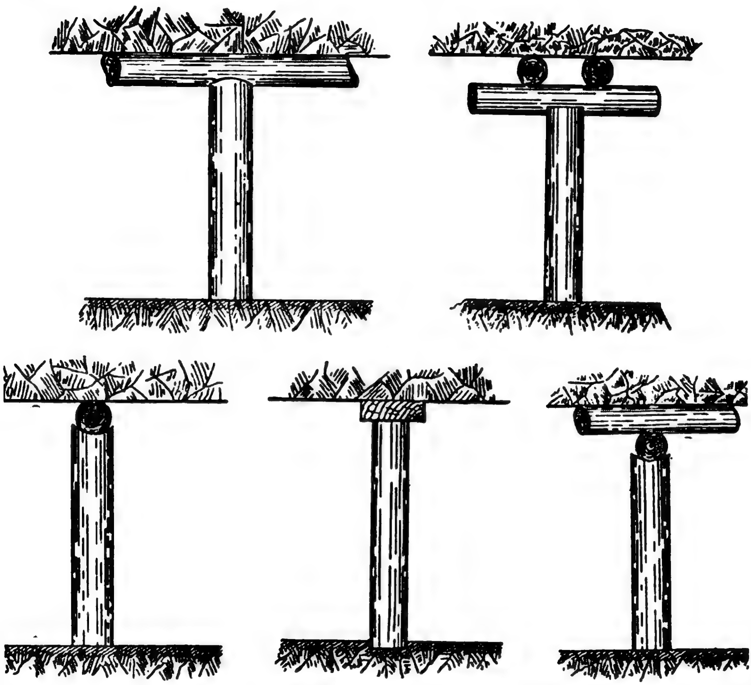
Any wood that is long-grained and elastic, and will yield to pressure or thrust, is suitable for mine timber, because props of such wood often serve the purpose for which they are used, even when partially fractured.

Methods of Support.—In timbering at the face, single props are usually set with a 'lid' or 'bonnet' on the top. The 'lid' may be either a square, half-round, or round piece of timber. Very often fractured props are sawn in pieces for this purpose. The lid prevents the weight from coming on the prop suddenly and fracturing it; it also gives additional resistance to the prop, and spreads the pressure over a larger area. In mines where the pavement (floor) is hard and slippery, the props are often pointed, so that they will crush up when the pressure comes on, and grip the floor, and thus be prevented from sliding away at the foot. Figs. 148 to 153 show some of the methods of setting single props adopted at the working face.

Wooden chocks or pillars are also largely used for supporting the roof in main roads, particularly in longwall workings where there is insufficient rubbish to completely pack the entire goaf.

These wood chocks are composed of either round or square pieces of timber, from 3 to 6 ft. long, and sometimes larger.

The pieces of timber are laid lengthwise horizontally, and another layer is laid on at right angles, this alternation being continued up to the roof, the topmost layer being driven tight with wedges.



FIGS. 148–153.—Methods of supporting roof.

The open spaces between the layers of timber are sometimes filled in with rubbish. If the chocks have to be withdrawn, as they often are in longwall faces, they are built on a layer of loose material, 3 or 4 in. thick, which facilitates their removal when required (fig. 154).

In setting props at the face it is unusual to set them at right angles to the inclination except in flat seams; the greater the inclination the more the props are set off the perpendicular, or 'underset.' The props when set should lie towards the rise of the seam, as this is found to give the best results. On the other hand, they should not be too much underset, as in such circumstances they

offer less resistance to the roof, while, as they also require to be longer, they are consequently weaker.

The amount of 'underset' or inclination given to the props ought to vary with the degree of dip, for props set with the same inclination in a steep seam, as in a comparatively flat one, would be apt to fall out before the pressure of the roof had tightened them.

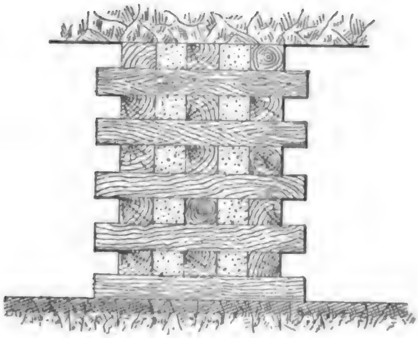


FIG. 154.—Wood chock.

Mr A. R. Sawyer gives the following table, which shows the maximum and minimum angles at which props should be set in varying inclinations.

Degree of inclination of seam.		Angle or 'underset' of props. Minimum.		Maximum.
6°	...	0°	...	1°
12°	...	0°	...	2°
18°	...	1°	...	3°
24°	...	1°	...	4°
30°	...	2°	...	5°
36°	...	2°	...	6°
42°	...	2°	...	7°
48°	...	3°	...	8°
54°	...	3°	...	9°

Timbering the roadways is practised in a variety of ways, according to the nature of the roof, floor, and sides. When the roof alone requires support and the sides are hard and firm, a cross-bar will be sufficient without it being supported by props. The cross-bar is fixed by 'needling' it into a hole *a* on one side and a tapered rest *b* on the other side for it to rest on, as in fig. 155; sometimes the bar rests on a prop at one end, and is 'needled' in at the other end, as in fig. 156.

When the sides alone cannot be depended on to support the cross-bar, a common arrangement is to put up a cross-bar or crown, either round or half-round, and to support it at each end by a prop set at a slight angle towards the centre of the road, so as to decrease the

span and strengthen the 'crown' (fig. 157). In this method nothing requires to be done to the timber beyond cutting it to the required length. Where heavy side pressure along with roof pressure has to be dealt with, various methods are used in which the timber is cut or notched in a special manner. That known as the Welsh system of timbering is much employed under such circumstances, and is found to give good results when the timber is properly put up. The sides are supported with 'lagging' of either square or round pieces of timber placed longitudinally behind the props. The 'sets' are often prepared at the surface and sent down ready for use. The advantages claimed

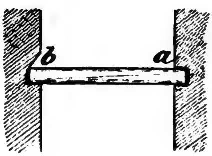
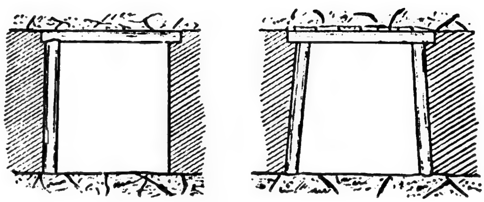


FIG. 155.—Cross-bar.



FIGS. 156, 157.—Cross-bar and prop.

for the Welsh system are: smaller cost than in using ordinary timber, as shorter crowns are employed and greater resistance to side pressure. On the other hand, the increased cost of preparing the wood and setting it must be taken into account.

When the roof and floor are both bad, it is very difficult to keep the roads in good condition by ordinary methods of timbering, and to prevent the floor from creeping, 'sill' pieces are often used, and the props notched into them as well as into the crown-pieces. These 'sill' pieces, however, have their disadvantages where the floor is given to creeping, for when the road requires adjustment, as it frequently does under such circumstances, the 'sill' pieces give trouble and often occasion the danger they were meant to avert.

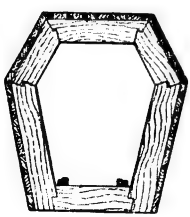
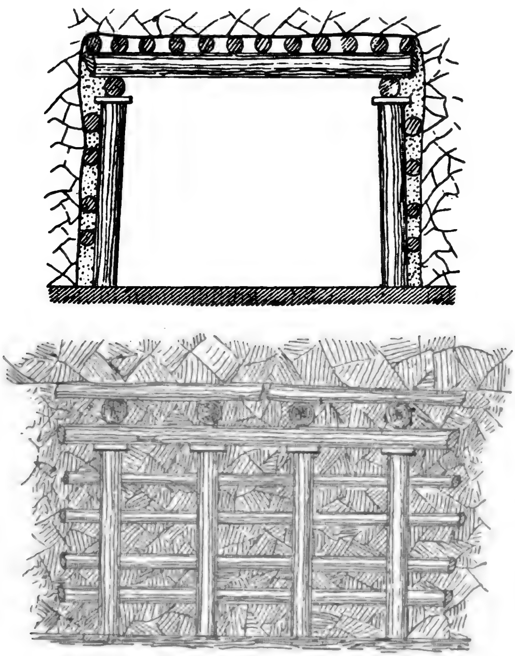


FIG. 158.—Wood set, as used at Comstock Mine.

At the great Comstock Mine, where enormous side and top pressures have to be withstood, a special system of timbering is adopted. Fig. 158 shows the construction of one of the sets employed. Each set consists of six pieces—a sill piece, top piece, and two pieces for each side. These pieces are usually cut square and notched in the manner shown. Between every two sets a close covering of lagging is laid all round, top and sides. While this system appears to be rather expensive, and more suited to rich metalliferous mines, cases have come under the writer's notice in which it might have been used with good effect in coal mines, as ordinary sets made of heavy

larch sometimes only lasted about two weeks, owing to the great side and top pressures.

When the roof is heavy, yet has no bed of hard rock in it, and the roads have to be driven comparatively wide, such as in pony roads and at branchings, it is often very difficult to properly support it. In such circumstances, only the best heavy larch should be used, both for crown-pieces and supports. Figs. 159, 160 show a method of supporting such roads. Crown-trees are set, and temporary props put up to the centre. Other crown-trees are placed at right angles to these, or parallel with the road along the ends of the cross-bars, and to those crowns parallel with the road the props are set, the whole being firmly fixed with lofting and wedges. Where roads branch off,



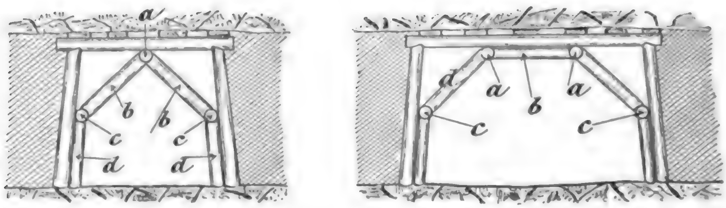
FIGS. 159, 160.—Supporting heavy roofs.

diagonal sets are sometimes erected to assist the cross-sets. Often when the road is very wide, and two lines of rails are used, it is a practice to set up centre props as well as end ones; but while this undoubtedly strengthens the crowns, it often becomes a source of danger, particularly where there is much traffic on the roads.

On the Continent, where great attention is paid to timbering, some peculiar methods are adopted for the support of roads. Figs. 161,

162 show two of these methods. In the former an ordinary set of timber is first placed in position, short props *d d* are then placed close against the main props, and on the top of these other pieces *c c* laid longitudinally and parallel with the roadway. From these longitudinal pieces *c c*, other posts *b b* are set at an angle so as to meet in the centre of the cross-bar, where another bar *a* is fixed in line with the road. On wide roads this system has the disadvantage of reducing the space considerably. To overcome this difficulty the method shown in fig. 162 is adopted.

Here an ordinary set is fixed as before, and the short pieces and the longitudinal pieces *c c* placed as already described. In the centre, and immediately against the crown-tree, is placed a shorter crown *b*, and against the ends of this short piece are two pieces *a a* placed at right angles or parallel with the roadway. From the longitudinal pieces *c c* on the top of the short props, posts are set at



FIGS. 161, 162.—Continental methods of timbering.

an angle to meet the pieces *a a*. By this method increased space is gained in the roadway.

In fixing the timber by either of these methods, the ordinary sets are first placed in position, and then all the auxiliary pieces are fastened to the former by thin wire until the set is completed, and when the pressure comes on the wire is of no more use. None of these methods has been adopted, to any extent, in this country, and it is questionable if it would not be cheaper to build side walls and use iron or steel girders under such circumstances.

Driving through Loose Ground.—When the roof or sides are very loose and have a tendency to 'spill' or run, a special method of timbering must be adopted.

The methods of securing such loose ground are shown in figs. 163 to 165.* An ordinary set is first placed in position with posts *a a* and crowns *c c*, and if the floor is very soft a sill-piece *d* is added.

Pieces of lagging are then inserted above the crown-trees, and driven in with the ends pointed upwards a few degrees off the horizontal. These continue to be driven forward as the work proceeds. Other lagging pieces are driven in behind the posts, and also inclined

* "Methods of Timbering," *Cal. State Mining Bureau Bulletin*, No. 2.

outwards at the end 10° to 15°, the side pressure gradually bringing them to bear closely against the props.

The two systems shown in figs. 163 to 165 are practically the same in principle, but differ materially in detail. In fig. 163 the lagging is inserted between the two crown-trees, which are separated by wedge-shaped blocks, one of which is placed at the centre and one at either end. The lagging is then driven forward, as already

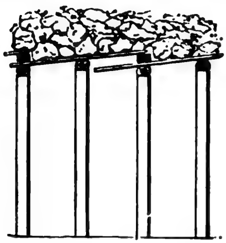


FIG. 163.

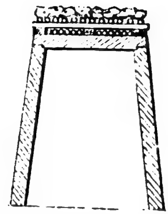


FIG. 164.

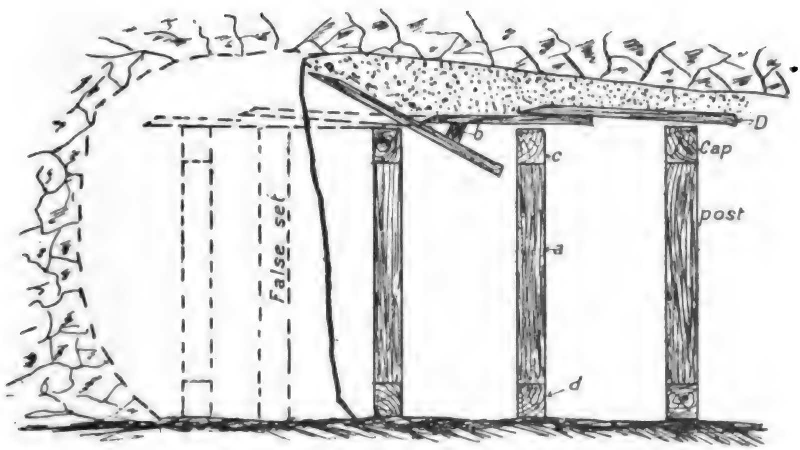


FIG. 165.—Timbering in loose ground.

described. If the ground is very heavy, a 'false set' is erected and the ends of the lagging rest upon it. As the excavation progresses the lagging is driven forward, until the further ends find a secure resting-place on the regular sets. The false set is then knocked out and the same operation repeated with the next set.

The only difference between these two methods is that in the one (fig. 163) there are two cross-bars—one light and one heavy; while in the other (fig. 165) the lagging is inserted beneath the advancing ends of the set next behind. In both methods the lagging is kept

pointed slightly upwards by the insertion of a block of wood *b*, which is placed between the portions already fixed and those being driven. When the lagging is driven forward a certain distance, this block is allowed to drop out.

To facilitate driving, the lagging is sharpened to a point at the one end. With these methods, even with the greatest care a quantity of the loose ground will run through and leave cavities behind.

Cost of Timber.—The cost of timbering varies greatly in different districts and at different collieries, and even in different sections of the same colliery, and may be anything between ½d. and 9d. per ton of coal raised. It will vary with the nature of the roof and floor and of the coal, the inclination of the seam and its depth from the surface, and also according to the method of working employed. The following table shows approximately the cost of timbering in a number of collieries in Great Britain.

TABLE SHOWING COST OF TIMBER PER TON OF COAL RAISED.

No. of Colliery	Thickness of Seam.	Depth from Surface.	Nature of Roof.	Inclination of Seam.	Method of Working.	Cost per Ton.
	Feet.	Feet.				Pence.
1	2½ to 3¼	1000	Fairly good	Comparatively flat.	Longwall	3
2	2 to 4	1050	,,	...	,,	2¾
3	2 to 5	1100	Soft roof	...	,,	4
4	1½ to 1¾	1550	Bad roof	1 : 5, and upwards	,,	8½
5	2½ to 4	100 to 250	Fairly good	Flat, to 1 : 15	Longwall and pillar and stall	2¾
6	5½	1050	Soft shale	1 : 9	Double stall	8
7	6¼	850	,,	Flat, to 1 : 14	Longwall	5
8	5¼	1200	Shale	1 : 20	,,	2½
9	7	1700	Good roof	Flat	Pillar and stall	1¾
10	5¾	1600	,,	1 : 16 to 1 : 18	,,	2½
11	6½	600	Bad roof	1 : 20	Longwall	3
12	5	1800	Good roof	1 : 12	,,	1
13	5	1200	Fairly good	...	,,	2
14	4	1150	Good roof	1 : 6	,,	½

In the Government collieries of the Saar district of Germany, the cost of timber is estimated to average 6d. per ton of coal raised.

Supporting the Main Roadways by Brickwork.—For main roads and pit bottoms, masonry is used to a very large extent, and probably no better method of securing a road has been devised. It is very much more expensive than timber, owing mainly to the extra cost for excavation, but where a roadway has to be used for a considerable number of years, it will in the end be the cheapest method. There are two principal methods of supporting roads with masonry, viz., by building perpendicular side walls to a certain height and then springing an arch for the roof, and by building the

masonry all round the roof, floor, and sides, no part of the walling being built perpendicular, but each part having a certain curvature. This latter method is undoubtedly the better of the two, and masonry so constructed is very much stronger, but it is also a great deal more expensive, and unless the road has to stand for many years, and a large output is expected from it, it is questionable if the expense of such a system of arching would be justified. For most ordinary purposes, unless the floor is very bad and given to creep, the common method of supporting roads by two short perpendicular walls and a top arch will be found quite efficient.

The brickwalling is put in 9 in. to 18 in. or 24 in. thick, and a space should be left between the walling and the strata, which should afterwards be filled in with fine ashes or sand, as this will greatly assist the arching when the pressure comes on it.

Iron or Steel Supports.—Within recent years the use of iron and steel supports for underground workings has greatly extended, and, in certain circumstances, it is to be recommended. It must, however, be remembered that the conditions under which steel or iron girders can be used underground are altogether different from the conditions under which the same materials can be used at the surface. In the latter case, all the conditions to be satisfied can be accurately predetermined, the size and resistance of any supports required being ascertained by calculation.

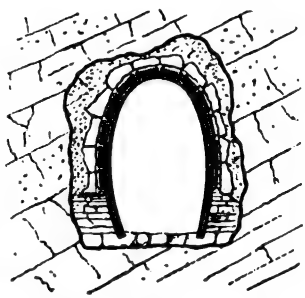


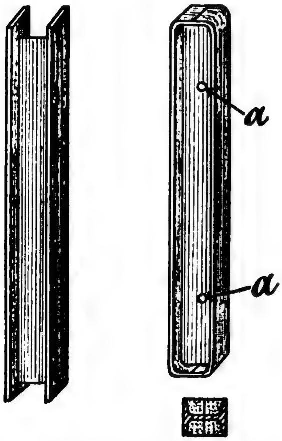
FIG. 166.—Steel supports.

Underground, these conditions can scarcely be ascertained at all, or, at least, only very partially; the top weight to be supported may be unknown, and further complications are introduced when heavy side pressure is encountered. Instead of the load being uniform, as on the surface, it is very varied, and the supports are subject to great and suddenly applied pressure. Steel girders, however, seldom break under sudden pressure or weight, but nearly always give indications of such pressure by deflecting in the centre. Girders have been known to show 5 to 7 in. of deflection under great top pressure before breaking.

In a large number of mines there are main haulage roads and horse roads where the strata have settled, and where the pressures are fairly uniform. In such roads, steel or iron supports can be used with advantage. Again, in return air-ways where the air is hot and foul, and contains a good deal of moisture, wood very rapidly decays and requires frequent renewal, and, in such circumstances, steel or iron supports may beneficially replace it.

Iron or Steel Props. — While iron or steel is better suited for

use as cross-bars or crowns, props made of these materials have also been used at the face of longwall workings. They are, of course, much more expensive than timber props, and it is therefore necessary that they should be used only where they can be withdrawn and re-set, otherwise the cost would be too great. Cast-iron props are not of much use, as they are heavy and easily broken.



Figs. 167, 168.—Steel pit props.

Steel girders, of H-section, present a somewhat sharp and uneven surface to the roof or to the timber lids when used. To overcome this difficulty Firth's patent prop is used.

A piece is cut out of the web at each end, and the top and bottom flanges turned over, which enables the ends to present a flat surface to the roof and floor. A hole *a a* is punched in the web, about a foot from each end, for the insertion of a hook to assist in withdrawing the prop (see figs. 167, 168).

Steel or Iron Sets.—These are used in some collieries in Great Britain, but they have a more extended use on the Continent, where, as already stated, more heed is paid to supports for roadways. Steel or iron sets are best adapted for main roads, particularly for roads where the strata have settled.

In France and Germany, a simple arrangement, shown in fig. 169, is used. Iron or steel bars, weighing 24 to 30 lbs. per yard, and shaped like girders, are bent in the form of a horse-shoe to suit the roadway and set up 2 to 3 ft. apart. The space between the webs is filled up with planking, 1½ in. to 2 in. thick, forming a neat strong lining to the road. This method is said to cost about £2 per yard for a road 7 ft. high and 6½ ft. wide at the bottom.

Another method, employed in the North of France, is to form the girder in two pieces, curved at the top and joined in the centre by two fish-plates fastened by four bolts. This method is best suited for heavy roofs and hard sides. Fig. 170 shows the detail of the fish-plates at the centre where fastened. The cost in this case is estimated at £5, 1s. 5d. per lineal yard for a road 8 ft. wide and 6½ ft. high.

In the Government Lead Mines, in the Hartz Mountains, the roadways are supported with flat-bottomed rails, 14½ lbs. per yard section. The ends of the iron arch are lodged in holes drilled in large stones (see fig. 171) set in the floor and fastened by wooden wedges or cement. Between these stones other stone blocks are

inserted, in order to keep them apart and thus ensure stability. The lining or lofting is carried out with the same kind of rails, each 19 ft. 8 in. long, arranged longitudinally, the flat bottoms being in contact with the base of the flat rails. The cost of supporting roadways in this manner is about £1, 13s. 2d. per yard. Supporting the same roads with masonry costs £2, 9s. 0d. per yard, and with timber £1, 2s. per yard (first cost only).*

When roof and floor are both soft, and the floor given to creeping,

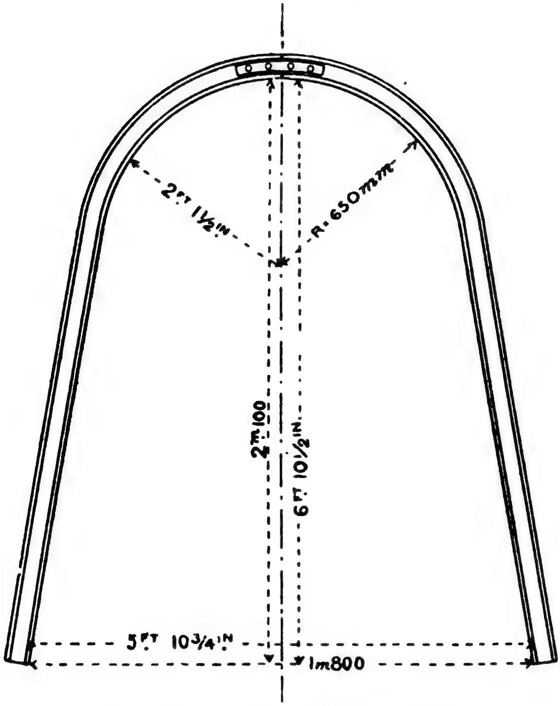


FIG. 169.—Iron sets for supporting roof.

the support is sometimes made in two or three pieces curved to suit the roadway and fastened at the joints by fish-plates and bolts as before (see fig. 172). A cast-iron sleeve is often used instead of the fish-plates. The sleeve is made to slip over the end of the girder, and when the pieces are fitted together it is drawn over the joint and fastened with wood keys or wedges. Fig. 173 shows the construction of such a sleeve.

At the Altenwald Coal Mines,† near Saarbrücken, iron supports

* *Trans. N. Eng. Min. and Mech. E.*, vol. xxxvii. p. 137. † *Ibid.*, p. 138.

are used in the form of an elliptic arch. To prevent the supports from shifting, horizontal props are inserted from arch to arch at the highest points. The plank covering is of oak, and each plank overlaps the other, to allow some play under heavy pressure. The cost for this kind of support was £2, 18s. 2d. per yard. Brickwork would not have been applicable in this case, owing to the continuous settling of the floor.

At the Nunnery Colliery, Sheffield,* the main roads were supported by steel girders, which were themselves supported on props of larch wood (see fig. 174). The girders were I-section, 4 in. wide, 5 in.

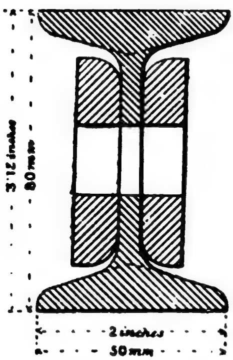


FIG. 170.—Details of fish-plate.

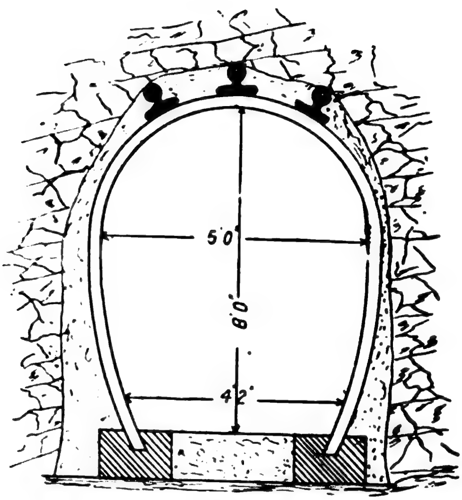


FIG. 171.

deep, with a web $\frac{3}{8}$ in. thick, and this was calculated to give the same support as a Norway larch beam 12 inches square.

The girders were supported on props of larch, 8 in. to 10 in. diameter, and the sets were put in about 3 ft. apart, with lagging above.

To prevent the girders from being pushed out at the top by side pressure, a lug or band of iron, $1\frac{3}{8} \times \frac{3}{4}$ in., was shrunk on at each end immediately in front of the prop.

In some collieries in Staffordshire, hollow cast-iron props are used. These props have a flange 8 in. diameter at top and 9 in. diameter at bottom (see fig. 176). A chair made for the purpose drops into the top of the iron column and receives a reversed iron rail weighing 50 lbs. per yard. These sets are placed 3 ft. apart and are lofted on top

* *Ore and Stone Mining*, Sir C. Le Neve Foster, Sixth Edition, p. 273.

with planks or rails, the spaces between these being tightly packed with stones. This method of support makes a capital roadway, but is best suited for roads where no great side pressures exist. The cost of this system is £2, 4s. per lineal yard in an average-sized roadway (see figs. 175, 176).

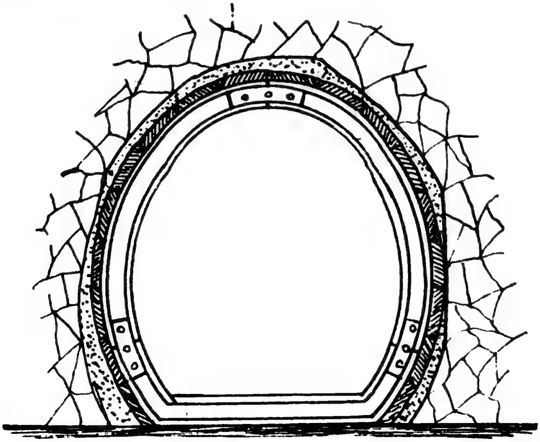


FIG. 172.

At St Helen's Colliery, Cumberland,* flat-bottomed steel rails were used to support a main haulage road at a depth of 170 fms. Posts of the same material were likewise employed to support the

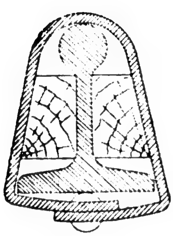


FIG. 173.—Construction of sleeve.

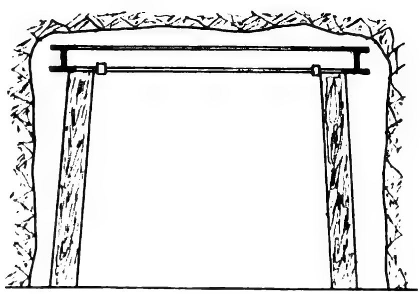
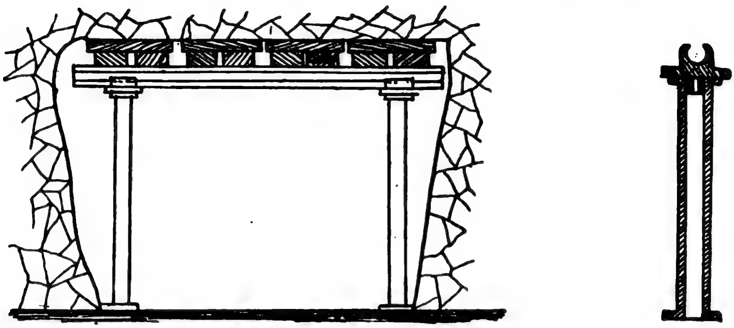


FIG. 174.

cross-pieces; figs. 177, 178 show how the rails were secured. The crowns were 10 ft. long and the supporting props 6 ft. The latter were set 6 in. off the perpendicular, and were cut and dressed so that

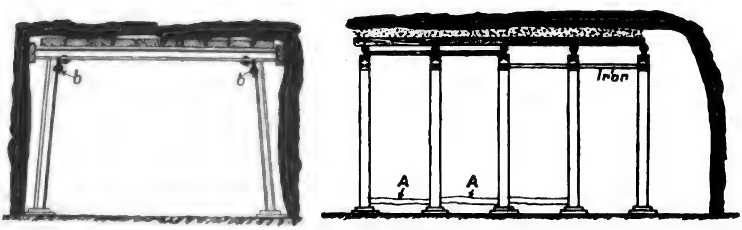
* *Trans. Min. Inst. Scot.*, vol. xiv. pp. 242–249.

the crown would rest evenly on the top. The lower end of the props rested in a cast-iron sole (fig. 179), arranged to give a solid foundation. Near the top of each prop, and also near the ends of each crown, were drilled two holes ¾ in. diameter. An angle iron *b* (fig. 177) was then riveted on the end of the crown so as to fit the upright to which it was bolted, and in this way the legs were fixed and prevented from being pushed out. The sets were placed 2 to 3 ft. between



FIGS. 175, 176.—Cast-iron props.

centres and lagged on the top with 3 in. planking, and also, when they could be had, with old rails. A piece of wood A A was fitted in between the uprights to further strengthen them. The rails used weighed 79 lbs. per yard, and cost 3s. 6d. per cwt. delivered; the cost for steel H-girders, weighing 54 lbs. per yard, would have been 5s. 9d. per cwt. The cost per lineal yard for this method of supporting the



FIGS. 177, 178.

roadway, including rails, labour, etc., was £1, 14s. 7d., while the cost of brick arching, 14 in. thick for the same road, was estimated to be £2, 5s. 2d. per lineal yard, showing a difference of 10s. 7d. per yard in favour of the rails.

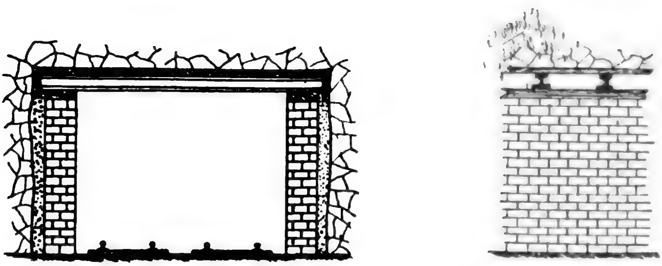
Securing Roads with Brick Walls and Girders.—Main haulage roads and shaft bottoms are often secured by building up a brick wall, 14 in. to 24 in. thick, on each side of the road, and stretching steel H-girders across them (figs. 180, 181). Along the top of the

wall is laid wood planking, 4 in. × 12 in., on which the girders rest. The wood helps to relieve any sudden pressure to which they may be subjected. The girders should be wedged tight, and a runner or strap of iron fixed between every two sets to prevent them from canting. On the top is placed a lagging of square or round timber laid close together, any spaces between being carefully packed. At Milnwood Colliery, Bellshill, the wood lagging was replaced by strips of iron about 3 in. broad and $\frac{5}{8}$ or $\frac{3}{4}$ in. thick. This system is pre-

Fig. 179.—Details of sole-piece.

ferable to using wood lagging, as the latter decays and requires frequent renewal. Sometimes sheets of iron, about $\frac{1}{8}$ in. thick, are stretched on the top of the girders, and wood lagging put above that; the sheet iron preventing the wood lagging from dropping on the road when it breaks or decays.

In another method of securing roadways by brick walls and girders,



Figs. 180, 181.—Brickwork and girders.

the latter, instead of being put in straight, are curved from the side of the wall on either side, which is said to increase their strength. It also gives increased height, but, of course, it is more expensive, as more rock requires to be excavated in the roof. The ends of the girders are laid on sheet iron, thus distributing the pressure over a greater area. The cost of this method is £13, 9s. 6d. per yard. The calculated cost for arching the same roadway was £16, 8s. 11d. per yard.

At Lanemark Colliery, New Cumnock, some rather ingenious

methods are adopted for supporting the roadways. On each side of the road, side walls are built from the rubbish got from the workings, cement, mixed with sand in the proportion of one to four, being used to bind the rubbish together. At intervals along the road, brick pillars are built up to the same height as the stone building. Along the top of the side walls planks are laid, and on the top of these light steel girders are stretched across the road to support the roof. Figs. 182, 183 show this method.

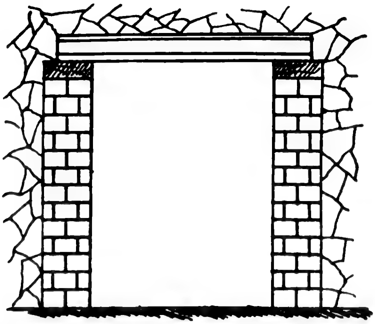


FIG. 182.—Cross-section.

Where the roof is fairly good and does not require any cross-girders, the roadways are secured as shown in figs. 184, 185. In this method the stone and cement walling is carried up to within 4 in. of the roof. In the space thus left, pieces of wood are placed lengthways, being wedged tightly to the roof. Between the stone and cement pillars a pillar of brick is built half-way up, and on the top of this a short prop is fixed between the pillar and the longitudinal planking.

Another variation in this system of supporting roadways is shown

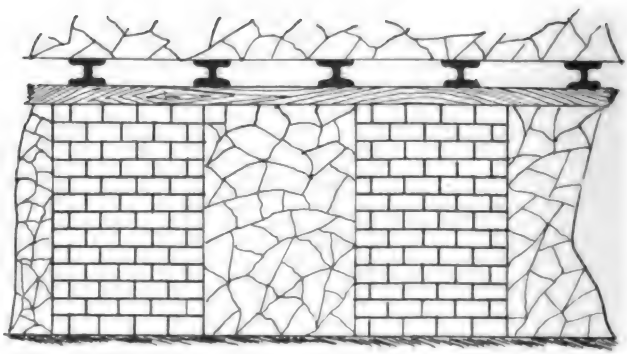


FIG. 183.—Longitudinal section.

in figs. 186, 187, in which a continuous wall of stone and cement is built up to about three-fourths the height of the road, and on the top of this building short pillars of brick and short props are carried up to support the wood. In both these latter methods no cross supports are used, the roof not requiring them. In other parts of

the workings the walls are entirely built of rubbish got from the working and cemented.

This kind of wall is found to act better in many respects than brick and lime. It often deflects to a considerable extent before giving way. The advantages claimed for such methods of supporting roadways are that only a very small number of bricks are required, compared with those in which masonry is used throughout, while walls of this description can be built very cheaply, ordinary workmen being able, without difficulty, to erect them. In pillar and stall working they also prevent slabs of coal from breaking off the pillars and falling on to the roadways,—an occurrence which often causes much inconvenience in workings, particularly on main haulage roads.

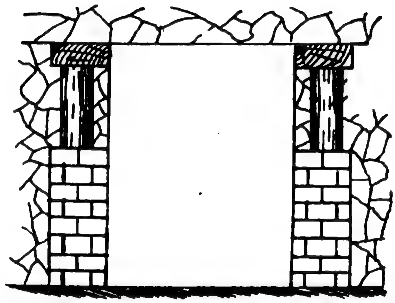


FIG. 184.—Cross-section.

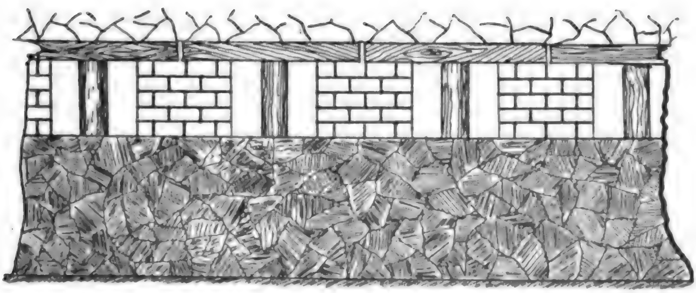


FIG. 185.—Longitudinal section.

The advantages of using brick wall and girders for supporting the roadways, instead of brick arching alone, may be stated as follows:—

Less space requires to be excavated for a given area, the saving in this respect being nearly 25 per cent.
Less time is required for erection, and hence less cost is incurred for labour.
Where the strata are soft, girders can be placed as the work proceeds, while with brick arching temporary supports would have to be used, thus increasing the cost.
Girders can be easily removed from one part of a mine to another and be used over again, whereas brickwork can seldom be removed.

The cost of iron or steel girders varies, and will depend to a certain extent on the proximity or otherwise of the colliery to iron-

works. In 1904 the cost of girders was about £5, 10s. to £6, 10s. per ton, and for the various sections in use, which are about 50, 66, and 72 lbs. per yard, about 9d., 1s., and 1s. 1d. per foot respectively.*

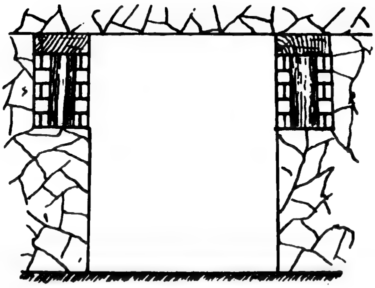


Fig. 186.—Cross-section.

Comparing the cost of supporting roadways with girders with the cost of timber for the same purpose, the first cost for girders will be, approximately, twice as expensive, but, on the other hand, they will last four to six times longer than the best wood, and will, as a rule, give a greater margin of safety.

Owing to the varying conditions in different mines, it is impossible to fix any definite weight or size of girder as being suitable for

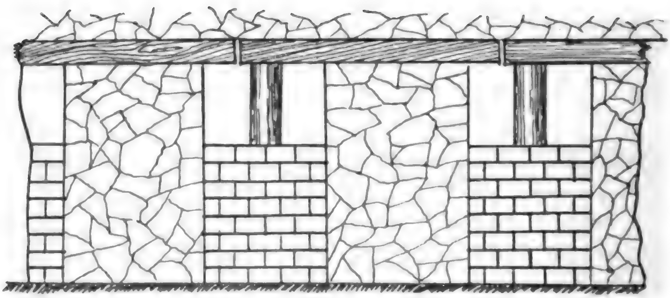


Fig. 187.—Longitudinal section.

a given span. Under comparatively equal loads,† however, the weights, dimensions, and safe loads for 8-ft. space girders are shown in the following table:—

Depth of Girder.	Width of Flange.	Thickness of Web.	Weight per Foot.	Estimated Safe Dead Distributed Load for 8 Feet Span.
Inches.	Inches.	Inches.	Pounds.	Tons.
5	4	$\frac{3}{8}$	$16\frac{2}{3}$	7
5	4	$\frac{1}{2}$	22	9
6	$4\frac{1}{2}$	$\frac{1}{2}$	24	12

* *N.B.*—The price of girders varies from time to time according to the market price of iron and steel.

† *Trans. I. M. E.*, vol. x. p. 274.

The following weights of girders are often used for different spans :—

Girders of $16\frac{3}{8}$ lbs. per lineal foot, in spans of	6 to 8 ft.	
,, 22 ,, ,, ,, ,,	8 to 10 ,,	
,, 24 ,, ,, ,, ,,	10 to 12 ,,	

The above can, however, only be taken as approximate sizes, and it would be best to err, if anything, on the safe side. In a colliery where the span was 16 ft., the girders used weighed 42 lbs. per lineal foot and were none too heavy.

Some of the advantages claimed for iron and steel props over timber are—

They are lighter and handier to work with than heavy wooden beams.
Girders give increased space for ventilation compared with timber.
There is no pollution of the air as is the case with decaying timber.
There is no risk of catching fire; which is so often the cause of underground fires where the timber is in a dry condition.

Strength of Timber.—The strength of timber is not always easily determined, and no definite rules can be given as to the size of props or crown-trees to be used underground. The circumstances prevailing in each colliery as to roof, floor, and sides, combined with everyday experience in practical working, seem about the safest and best guides to depend upon.

Props set in the workings may be said to break in three different ways, viz.: (*a*) by fracture or 'buckling' alone; (*b*) by buckling and crushing combined; (*c*) by crushing alone. In the first case, props generally give way when their length is from twenty to thirty times their diameter, in the second when their length is from ten to twenty diameters, and in the third case when the length is under ten diameters.

From numerous experiments on the strength of pillars (or props) of timber, the following law has been deduced :—

"*The strength of pillars of timber of equal sectional areas is inversely proportional to the square of the length.*"

Thus, with lengths in the ratio of 2, 4, and 8 ft. the strength will be in the ratio of $(\frac{1}{2})^2 : (\frac{1}{4})^2 : (\frac{1}{8})^2$, or $\frac{1}{4}$, $\frac{1}{16}$, and $\frac{1}{64}$.

Taking pillars of the same sectional area, one 2 ft. in length has sixteen times the strength of one 8 ft. in length and four times that of one 4 ft. in length. This is well known in everyday practice, and the props are usually increased in diameter, according as the height of the working increases. For ordinary large props the crushing strain is about $1\frac{1}{2}$ to 2 tons per square inch, according to the age of the wood and the seasoning it has undergone.*

If timber be cut when green, and allowed to season or dry gradually, it is found to gain in durability, as was proved by the experiments carried out by Professor Louis,† who records a gain of as much as 49 per cent. in the strength due to seasoning in ordinary pit props.

* *Trans. I. M. E.*, vol. xv. p. 352. † *Ibid.*, p. 354.

This fact is fully recognised on the Continent and at many collieries in France and Germany; the props are thoroughly seasoned in specially-constructed drying sheds before they are used underground; and in some cases they are seasoned artificially by an electric process which is said to give good results.

In a number of tests which were recently carried out by the Government colliery officials in the Saar district, * the same results were obtained. The following table shows the effect of seasoning, as ascertained experimentally, on four different kinds of wood:—

Kind of Timber.	Wood shortly after Felling.		Wood Five Months after Felling.		Props artificially dried in a Temperature of 149° Fahr.	
	Resistance to Compression. Lbs. per Sq. Inch.	Specific Gravity.	Resistance to Compression. Lbs. per Sq. Inch.	Specific Gravity.	Resistance to Compression. Lbs. per Sq. Inch.	Specific Gravity.
Beech, with bark .	3243	1084	3570	1094	3627	0915
Fir do. .	2802	0885	3044	0845	3385	0656
Pine do. .	2631	0984	2716	0917	2958	0647
Oak do. .	2475	1235	2133	1050	2958	0825

It must not be forgotten that the different specimens tested were special pieces, free from blemishes, and having little resemblance to the ordinary pit prop.

The following tables † give the crushing and tensile strains of various kinds of wood:—

Strength of Timber to Resist Crushing-Strains in Pounds and Tons per Square Inch.

Kind of Timber.	Maximum Dry.	Minimum Ordinary State.	Mean.	
	Pounds.	Pounds.	Pounds.	Tons.
Ash	9363	8683	9023	4·03
Beech	9363	7733	8548	3·81
Birch (English) . .	6402	3297	4850	2·16
Elm	10331	7950	9140	4·08
Fir (spruce) . . .	6819	6499	6659	2·97
Oak (English) . .	10058	6484	8271	3·69
,, (Quebec) . . .	5982	4231	5106	2·28
Pine (pitch) . . .	6790	6790	6790	3·03
,, (red) . . .	7518	5395	6457	2·88
Larch	5568	3201	4385	1·96

* *Glückauf*, Berginspektor Ch. Düthing, 1898, vol. xxxiv. p. 797.

† *Treatise on the Strength of Materials*, by Mr Thomas Box, 1883, p. 91.

Showing Strength of Timber to Resist Tensile Strain in Pounds and Tons per Square Inch.

Kind of Timber.	Maximum.	Minimum.	Mean.	
	Pounds.	Pounds.	Pounds.	Tons.
Ash	17,850	15,784	17,077	7·6
Beech	11,326	11,388	11,467	5·1
Birch	...	...	15,000	6·7
Elm	...	...	13,490	6·0
Fir	13,448	11,000	12,203	5·5
Oak (English)	15,500	13,620	14,560	6·5
Pine (Russia)	...	...	13,300	5·9
,, (Norway)	14,300	12,400	13,350	6·0
Larch	...	...	9,632	4·3

Showing Specific Gravity and Weight of Materials (Water at 62° Fahr. being equal to Unity).

Material.	Specific Gravity.	Weight of 1 Cubic Foot.	Measurement of 1 Ton.
		Pounds.	Cubic Feet.
Wrought iron	7·788	485·30	4·615
Cast iron (British)	7·087	441·60	5·070
Oak (seasoned)	0·777	48·42	46·260
Elm	0·588	36·65	61·130
Pine (yellow), seasoned	0·483	30·10	74·410

The following rules apply to bars or beams of timber:—

1. The strength of bars or crowns of the same sectional areas is in direct proportion to their width. Thus a bar 12 in. wide is twice as strong as one 6 in. wide if both have the same thickness.

2. The strength of rectangular beams of the same length and width is directly proportional to the square of their depth ($W \propto d^2$); thus if two bars are of equal width, but one is 6 in. deep and the other 3 in. deep, their strength will be in the proportion of $3^2 : 6^2$ or $9 : 36$ or $1 : 4$, *i.e.* the prop 6 in. deep will have four times the strength of one 3 in. deep.

3. The strength of bars of equal sectional area varies inversely as their lengths ($W \propto \frac{1}{l}$). Thus a bar 12 ft. long will only have half the strength of one 6 ft. long, the sectional area being equal in both.

Formulæ for Strength of Beams of Timber.—Different formulæ are given by different authorities for finding the sizes and breaking weights of beams of timber, all of them giving slightly different results. But it must be remembered that most of the formulæ applied in engineering give only approximate results, and are not meant to be absolutely correct, as is the case with the formulæ employed in problems in pure mathematics. In engineering, materials for construction are generally allowed a large marginal factor of

safety, and there is, therefore, not the same necessity for very fine theoretical calculations.

Let L = length of beam or span in inches
B = breadth
D = depth
W = breaking load in tons
K = coefficient of rupture

The value of K for different materials has been found by experiment, and is given below :—

Wrought iron = 3·40	Beech = 0·65
Cast iron = 2·30	Fir = 0·60
Ash = 0·95	Oak = 0·75
Pitch Pine = 0·75	Larch = 0·75

(1) Both ends *supported* and beam loaded in the centre. $W = 4K\frac{BD^2}{L}$.

(2) Both ends *fixed* and load in the centre. $W = 6K\frac{BD^2}{L}$.

(3) Both ends *supported* and load evenly distributed. $W = 8K\frac{BD^2}{L}$.

(4) Both ends *fixed* and load evenly distributed. $W = 12K\frac{BD^2}{L}$.

In circular beams of radius R, substitute $4·7R^3$ for BD^2 in the above formulæ. These rules are very difficult of application to mine timber, as the load is very rarely either at the centre of the beam or evenly distributed along its length, and there is also the side pressure to take into consideration, which, in underground timbering, is often greater than that from the top, and can never be accurately measured. The pressure per square inch due to the weight of the overlying strata alone would be equal to $D \times ·434 \times B$; where D = depth or thickness of overlying strata, B = specific gravity of strata, and ·434 = a constant number (average specific gravity of strata is about 2·5, taking water as unity or 1).

Strength of steel or wrought-iron girders of H-section :—

When W = breaking load in tons,
A = area of one flange (either top or bottom) in square inches,
f = tensile strength of material in tons per square inch (generally from 22 to 28 for steel and 18 to 20 for wrought iron),
D = depth of girder in inches (including both flanges),
L = length of span in inches.

Then $W = 4f\frac{AD}{L}$.

The safe load is generally taken at $\frac{1}{5}$th to $\frac{1}{7}$th of the breaking load for steel girders.

EXAMPLES.

(1) Find the breaking weight at centre of a pitch pine beam, 12 in. deep, 8 in. broad, and 18 ft. between the supports, ends fixed and load in centre; also find

the depth of beam required for a breaking load of 24,000 lbs., if the width of beam is 6 in. and the distance between the supports 12 ft., load uniformly distributed.

Suppose the beam to be fixed at both ends, then

(a) $W = 6K\frac{BD^2}{L}$

$= 6 \times \cdot 75 \frac{8 \times 12^2}{18 \times 12}$

$= \frac{6 \times \cdot 75 \times 8 \times 144}{18 \times 12}$

$\therefore$ W = 24 tons

(b) $W = 12K\frac{BD^2}{L}$

$\frac{24000}{2240} = 12 \times \cdot 75 \frac{6 \times D^2}{12 \times 12}$

$\frac{24000}{2240} = \frac{12 \times \cdot 75 \times 6 \times D^2}{12 \times 12}$

$\therefore 112 \times 1 \times 0 \cdot 75 \times 1 \times D^2 = 2400 \times 1 \times 1.$

The breaking strain is, therefore, 24 tons.

$$\therefore D^2 = \frac{\cdot 2400 \times 1}{112 \times \cdot 75} \qquad \therefore D = \sqrt{28 \cdot 57} = 5 \cdot 3 \text{ in.}$$

The depth of beam required would be 5·3 in.

(2) If a beam 10 in. broad and 14 in. deep has a breaking strain of 30 tons, what length of span would it support under a maximum load, supposing the beam to be simply supported and the load to be in the centre?

$$\text{Here } W = 4K\frac{BD^2}{L},$$

$$\therefore 30 = 4 \times \cdot 75 \frac{10 \times 14^2}{L}, \text{ and } 30\, L = 4 \times \cdot 75 \times 10 \times 196.$$

$$\therefore L = \frac{4 \times \cdot 75 \times 10 \times 196}{30} = 196 \text{ in. or } 16 \cdot 3 \text{ ft.}$$

(3) What would be the breaking load of a wrought-iron girder of H-section, with top flange 4½ in. broad, depth of girder 6 in., and span between supports 10 ft.?

Taking the tensile strength of wrought iron at 20 tons,

$$W = 4f\frac{AD}{L} = 4 \times 20 \frac{4 \cdot 5 \times 4 \cdot 5 \times 6}{10 \times 12} = 81 \text{ tons.}$$

Preservation of Timber.—Timber required for use underground, or, indeed, anywhere, should be felled during the winter when it has but little sap in it, because sap in wood ferments and produces rapid decay. It should also be well seasoned before being used, and if these two points are carefully attended to they are frequently all the timber requires to preserve it. The bark should also be removed before the timber is used underground; if this is done, there is less liability to decay, and when this does set in it is easier detected.

Various methods of preventing dry rot have been tried. Good ventilation is necessary, as timber decays much faster in foul, hot air than in a pure atmosphere. Water is also a good preservative, and in some places the shaft timber is kept wet for this purpose. The water acts by washing off the spores of the fungi as fast as they are formed.

The various methods of preserving timber :—

By common salt dissolved in water.
By impregnation with metallic salts, such as sulphates of copper and iron, chlorides of zinc or magnesium, etc.
By the use of creosote.
By coating with tar, etc.

Timber is often treated with brine made with common salt, in the proportion of 1 lb. of salt to four or five gallons of water, the timber being allowed to get thoroughly soaked with the solution. It has the advantage of being cheap and easily applied. Sulphate of iron is also much used, and has the recommendation of being effective and economical.

Chlorides of magnesium and zinc are used for preserving timber. In the zinc process, a solution of chloride of zinc is forced under pressure into the timber. The solution consists of one part of liquid chloride of zinc (specific gravity of 1·5) mixed with 35 gallons of water. One gallon of this solution weighs 15 lbs. and contains about 3½ lbs. of metallic zinc. This process is said to make the wood firm, hard, and proof against the attacks of insects and dry rot.

Aitken Process.—In this process the timber is soaked in boiling water containing a strong solution of common salt and chloride of magnesium. The proportion of common salt to the latter should be 7 to 1, and a certain proportion of undissolved salt requires to be kept at the bottom of the tank used for steeping. The timber treated should be free from bark, well seasoned, and thoroughly dry. The plant used at the Niddrie Collieries, near Edinburgh, where this system is in operation, consists of two rectangular iron tanks made of ½-in. boiler plate, 19 ft. long, 4 ft. wide, and 3 ft. deep, built into a brick seating with a hearth beneath. The boilers are fired with dross, and the tanks have a covering of loose boards.

The props are boiled for forty-eight hours; pitch pine and larch require longer treatment than softer woods. When the timber is removed from the tanks, it is stacked in a covered shed with free access of air, to dry, as it is quite soft and not fit for immediate use.

Cost of using Preparation.—With the above plant 15 tons of timber can be treated weekly at a cost of £2, 12s. 8d., or about 3s. 6d. per ton, which represents about 1s. 5d. per 100 ft. of 6-in. diameter prop wood. The plant itself costs about £100.

The process is said to make the timber brittle, and when it is used as 'sleepers' on roadways the nails do not hold well, owing to the oxidation occasioned by the salts present. To overcome this difficulty galvanised nails should be used. This method of treating timber is employed at a number of collieries in Scotland, among which are the Cadzow Collieries, Hamilton; Auchinraith Colliery, Blantyre; the Leven and Lochore Collieries, Fife; and others.

Creosote Methods.—Impregnating timber with crude creosote, which was first tried in 1842, is one of the best methods of preserving

timber, but such timber has the great disadvantage, particularly for mine work, of being very readily ignited, and is, therefore, less suitable for underground work than for other purposes. For railway sleepers at the surface, and even underground, where no danger of fire exists, creosoting adds greatly to the 'life' of the wood.

The effects of creosote are threefold: (1) it fills the pores and prevents saturation with water; (2) it destroys organic life; (3) the carbonic acid coagulates the albuminoids present in the wood and prevents decay.

Coal Tar.—Painting the timber with liquid tar is sometimes resorted to, but this also confers the disadvantage of being easily ignited.

Painting the props with ordinary whitewash is also a plan adopted, and one which gives fairly good results. While preservatives undoubtedly prolong the life of timber in underground workings, they seem at the same time to decrease its strength to a considerable extent. Professor Louis has made a number of experiments,* which show that timber thoroughly creosoted was diminished in strength to the extent of 8·5 per cent., while woods treated by the Aitken process were weakened from 8 to 20 per cent., according to the kind of timber treated.

The following table † shows the results of tests made at Saint Eloy, on the relative duration of differently preserved woods (unpreserved wood = 30) in France, upon different methods of treating oak, fir, pine, beech, birch, and poplar woods. Two specimens out of every ten experimented on were used in the natural state. The others were treated with solutions of (1) tar, (2) chloride of zinc, (3) sulphate of copper, (4) sulphate of iron, and (5) creosote, respectively.

Name of Preservative.	Name of Wood.					
	Oak.	Fir.	Pine.	Beech.	Birch.	Poplar.
Tar, ...	28·7	263·5	87·5	105·4	26·2	150·5
Chloride of zinc, . .	10·5	50·0	26·3	18·6	52·5	34·7
Sulphate of copper, .	42·1	12·0	8·0	1·8	2·5	15·5
Sulphate of iron, . .	18·0	12·5	4·2	4·7	3·7	2·9
Creosote, ...	1·7	2·5	4·4	0·6	3·3	1·3

Solutions of molasses have also been used successfully on the Continent and elsewhere.

* *Trans. Inst. M. E.*, vol. xv. p. 352.

† *Comptes-rendus mensuel des Réunions de la Société de l'Industrie Minérale*, 1890, p. 225.

CHAPTER X.

WINDING COAL.

Preliminary.—The operation of winding or raising the coal from the underground workings to the surface is one of the most important parts of the daily work of a colliery, for, in many cases, the output is limited only by the means available for raising the coal. When once the winding machinery is erected, it is clear that whatever the demands may become, the quantity of coal raised per day cannot exceed the capabilities of the machinery or the winding power. It follows, therefore, that what may be termed increased cost in the winding gear is of very small importance, when compared to the great advantage that may accrue from having, what may appear at the time, superfluous power which can be employed in case of need and if the output is capable of extension. All other surface arrangements must be subsidiary to the necessity of dealing effectively with the coal when drawn, otherwise much vexatious expense and delay will be entailed.

Pit-head Frames.—Pit-head frames were at one time almost entirely constructed of wood, but of recent years wrought iron and steel have been extensively used in their construction. Where a frame has to stand for thirty to fifty years, or possibly longer, it is a matter of economy to adopt iron or steel structures, as they are more stable and are not liable to decay like wood frames. For high frames, and for the heavy loads now raised at modern collieries, it is almost imperative to build the frames of steel. In cases where timber is employed pitch pine is generally selected, the size of the wood depending upon the height of the frame and the load to be raised. The following sizes are often used in practice:—for frames 20 to 30 ft. high, front stays and main supports 10 to 12 in. square; for frames 30 to 40 ft. high, front stays and main supports 12 to 14 in. square; and for frames 40 to 60 ft. high, the whole of the wood would be 14 in. to 18 in. square.

Pit-head frames are usually of two kinds, single and double, both sorts being largely used, according to the preference of those erecting them and the class of work for which they are designed. For heavy

loads and where pumping is required and tackling has to be fixed to the frame, the double type of frame is most suitable; a further

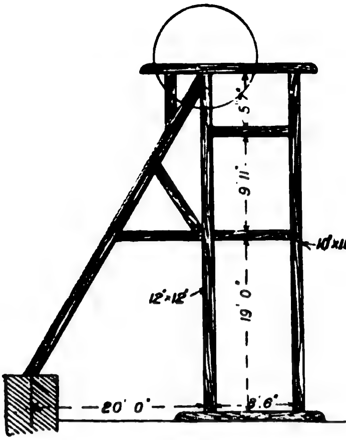


Fig. 188.—Side elevation.

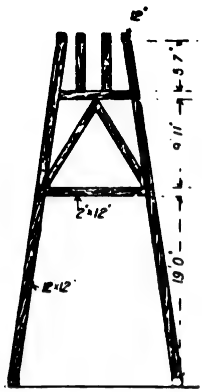


Fig. 189.—Front stays.

advantage being that pulleys for haulage ropes can easily be erected on them, so saving the erection of another frame for them

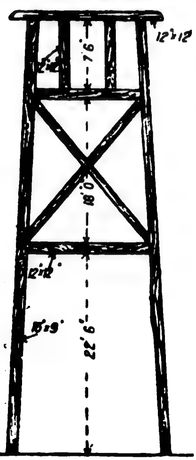


Fig. 190.—Back stays.

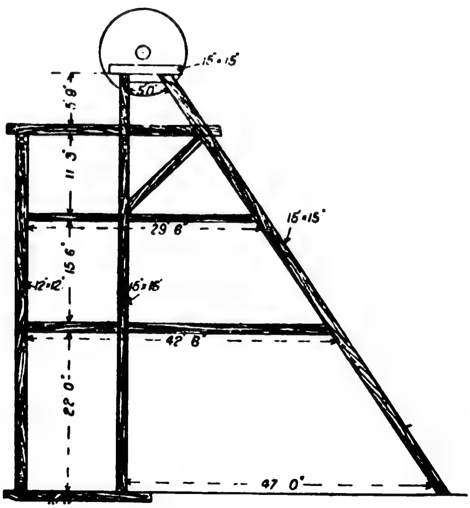


Fig. 191.—Side elevation.

close to the pit mouth where the room can often be ill spared. Where no pumping is necessary, a good single frame is, on the other

hand, just as good for winding, is neater, more easily erected, and is less expensive than a double one.

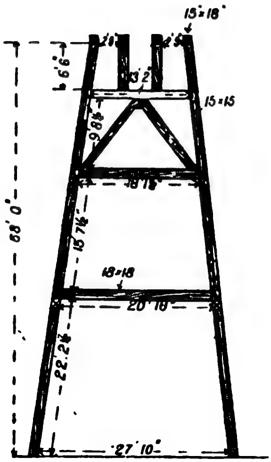


FIG. 192.—Front stays.

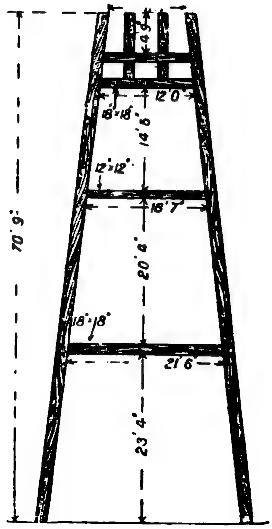
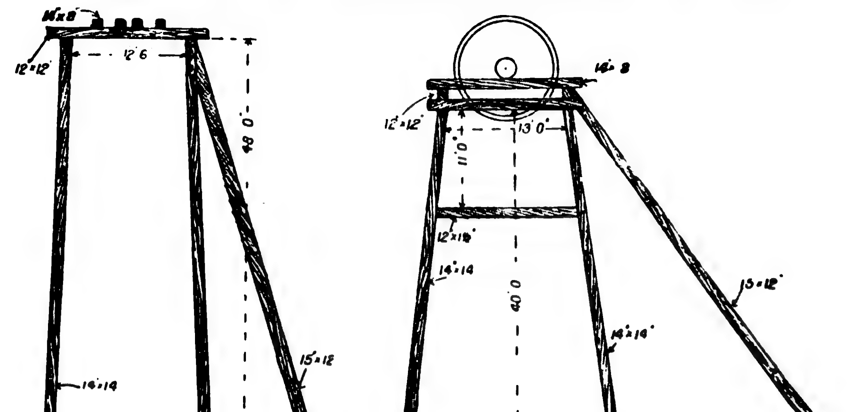


FIG. 193.—Back stays.

Figs. 188–193 show two good types of double wood frames.



FIGS. 194, 195.—Pit-frames and pit head-gear.

Figs. 194, 195 show a type much in use at large collieries in

Scotland, and known locally as a 'table frame.' Where much pumping is done, and block-and-tackle pulleys have often to be suspended from the frame, or, as is the practice at some collieries for the pulleys for haulage ropes, as well as the winding pulleys, to be placed on the frame; then this type of table frame is possibly the best form that can be adopted.

The whole construction should be firmly and carefully put together by careful and experienced workmen, and the parts fitted together previous to its erection, so that every part fits exactly. The cross-stays should be morticed into the uprights, about 3 in. being generally allowed, all the parts being well bound to each other by good, strong wrought-iron glands and plates. Fig. 196 shows the details of these glands and the manner of fixing. The wood in the frame should be smoothly planed, and two or three good coats of paint should be applied to preserve it from the weather. It should also be repainted every second year at least; this will prevent decay setting in.

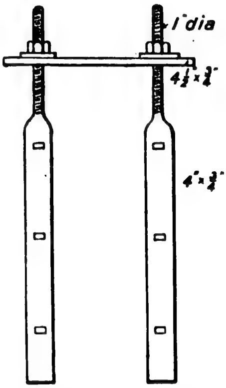


FIG. 196.—Manner of fixing glands.

Generally, the back and front stays are fixed at the bottom on sole-pieces running across the front of the pit and at right angles to the end of the back stays. These sole-pieces should rest on a good foundation of brick or concrete above the surface of the ground, to prevent moist earth from coming in contact with the wood, which will help greatly to prevent decay. Probably the best way is, however, to omit these sole-pieces and to fix the ends of both back and front stays into cast-iron shoes which rest on, and are firmly bolted to, pillars of masonry or concrete. The seam at the top of the shoe should be well filled with putty to prevent water lodging, otherwise this method of fixing is of little advantage so far as the prevention of decay is concerned.

The position of the back stays in regard to those in front is a very important consideration, as it is on this part of the frame the tension due to the winding ropes exerts itself. The back stays ought to be put up with a fairly large angle, otherwise the frame is liable to be drawn over by the tension or pull on the ropes; on the other hand, they ought not to be erected with too large an angle, otherwise their own weight will exert a pressure on the front stays and tend to push the latter out of position.

A good plan is to make the distance between the centre of the shaft and the foot of the back stays about equal in length to the height of the frame, or even longer; or else the distance should equal the height of the frame multiplied by the sine of the angle

made by the ropes with the pulley and drum. Suppose the pit-head frame is 60 ft. high, and that the angle that the ropes make between the drum and pulley is 50°, then 60 × sine 50° (0·7660) = 45·96 ft., the distance the back stay ought to be from the centre of the shaft. The position for the back stays may also be found graphically by employing the principle of the parallelogram of forces. Let *x y* be the ground line (fig. 197), *d* the position of the drum, and *p* the position of the pit-head pulley. Draw a line *a d* between these two, and another line *a c* representing the part of the rope hanging in the shaft to which the load is attached. Ascertain what the total load to be raised amounts to. Now, along the lines *a d*

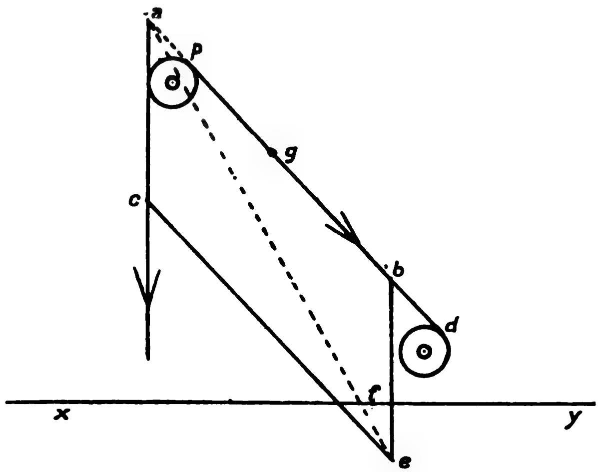


FIG. 197.

and *a c*, two forces which are equal and opposite to each other will be acting, the force along *a d* being that due to the pull of the engine required to raise the load, while the force along the line *a c* will be exerted by the load itself pulling in a downward direction. With any suitable scale, say 1 in. to represent 1 ton of load, lay off the distance *a c* equal to the total load, and along the line *a d* lay off the same distance *a g*; but to allow for contingencies, such as undue strain due to an over-wind, wind pressure, etc., it is better to make the distance *a b* along the line *a d* equal to twice *a c*. From the point *b* draw a line *b e* parallel to *a c*, and another line *c e* parallel to *a b*; a line joining the points *a e* represents the resultant of the forces *a c* and *a b*, and the point *f* where it cuts the ground line will be the position for the back stay.

Iron or Steel Frames.—As already stated, pit-head frames are often constructed of iron or steel. On the Continent the frames are

sometimes made of tubular material, but this type of frame has never come much into use in Britain; those most generally employed being constructed either on the lattice girder principle or of angle iron in conjunction with box girders.

Figs. 198, 199 show a frame mainly made in this way, 70 ft. high, which is less expensive than a lattice girder frame. The cost of such a frame would be about £350, including erection. At Palace

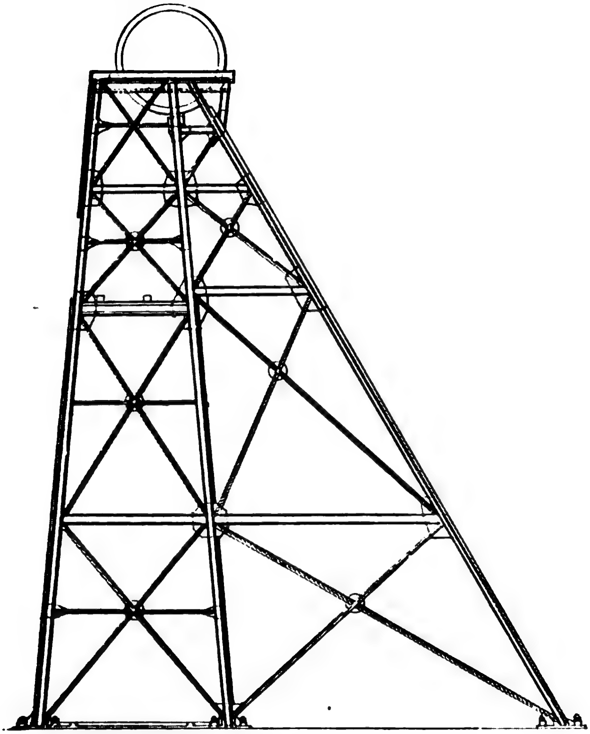


FIG. 198.—Steel frame (side elevation).

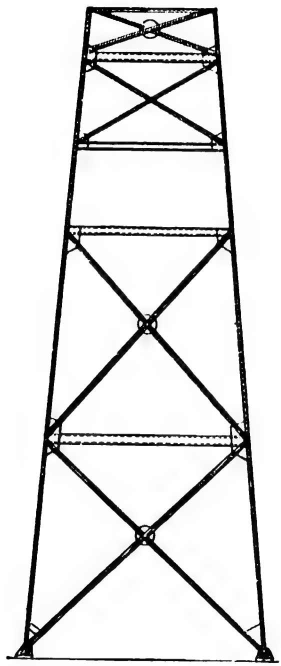


FIG. 199.—Steel frame (back stays).

Colliery and Bent Colliery, Hamilton, the principal parts of the frames are constructed of ordinary railway rails and lattice work on the back stays. Figs. 200–203 show this class of frame, which makes both a neat and strong erection.

Winding Engines.—Winding engines may be divided into two classes, viz.: (1) Direct-acting coupled engines; (2) non-direct-acting geared engines, either of which may be horizontal or vertical. The best type of winding machinery is a pair of coupled direct-acting

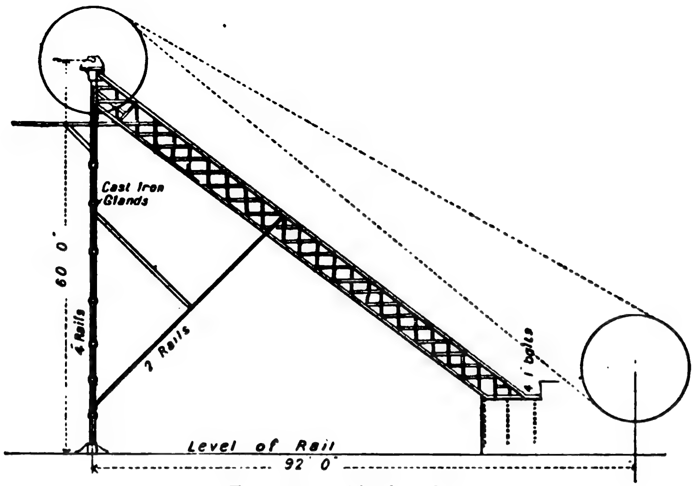


FIG. 200.—Side elevation.

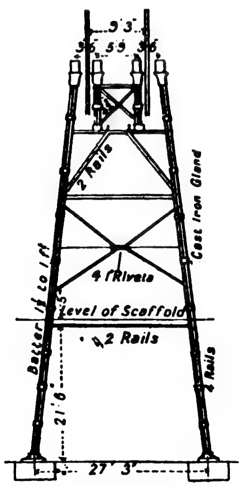


FIG. 201.—Front stays.

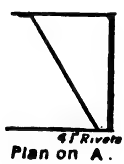


FIG. 203.

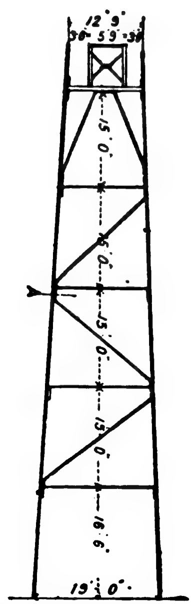
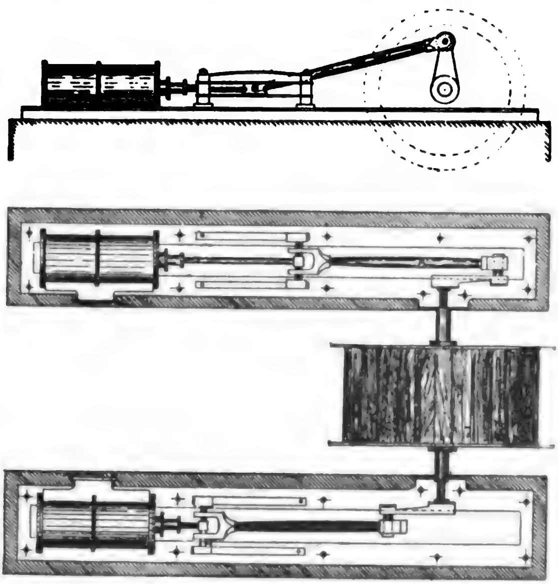


FIG. 202.—Back stays.

engines placed horizontally, as they are efficient, compact, easily cleaned and repaired, and well in view of the engineman. Figs. 204, 205 show the general arrangement of a pair of horizontal direct-acting engines.

Condensing and expansion forms of winding engines have not been much used owing to their difficulty of application for colliery work, the rapid winding and frequent starting and stopping being against their working economically. At a few collieries, however, they have been employed with fairly good results.

Coupled engines working at high pressure and provided with



FIGS. 204, 205.—Horizontal engines, with both cranks shown in position at end of stroke.

automatic cut-off valves are possibly the most efficient and economical type of machine for winding coal, as their working parts are few and not so complicated as in compound condensing engines.

Single-acting engines may be employed for winding small outputs from shallow shafts, if geared and fitted with a heavy fly-wheel. Such engines are not, however, to be recommended, as they are very unsteady in their motion, and occasion delay and annoyance when they stop on a 'dead centre.'

The following conditions should be satisfied in a good winding engine :—

It should be as direct-acting as possible, *i.e.* the connecting parts between the piston and the crank shaft should be few in number, as each part entails a waste of power.

The moving parts should be strong to resist stresses, and at the same time light enough to offer no undue resistance to motion. Parts moving upon each other should be carefully and smoothly machined in order to reduce friction to a minimum.

The steam should reach the cylinder easily at the proper time, and should also be able to leave the cylinder as easily.

The engine should be capable of being readily and immediately stopped, started, or reversed.

Speed of Engine.—The speed of winding engines varies according to the size and class of engine, but 250 to 400 ft. per minute, as the rate of piston travel, is considered good speed for winding engines.

Position of Winding Engine.—The laying down of the winding engine in a proper position is a very important matter. The exact site will, of course, depend on the position of the winding drum, as the engines should be placed at a suitable distance back from the shaft to afford sufficient inclination for the ropes from the pit-head pulleys to the drum.

The distance that the drum should be from the centre of the shaft

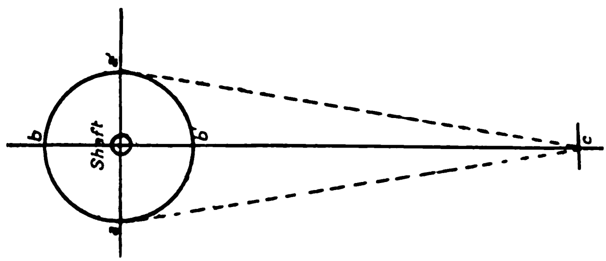


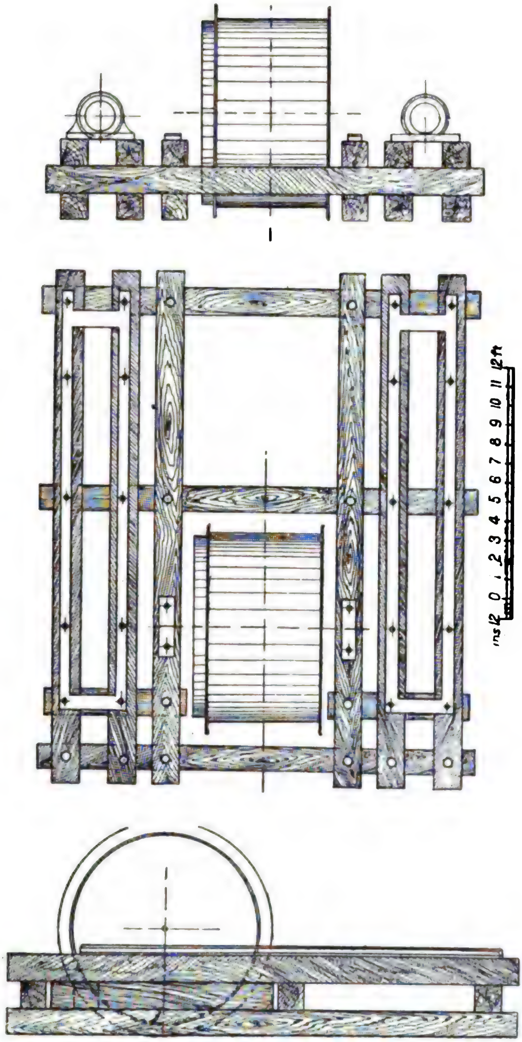
Fig. 206.

is equal to about 1 to 1½ times the height of the pit-head frame, which will give a fairly good angle for the ropes to work at. Sometimes the position of the drum is fixed so as to give the winding ropes an inclination of 45° with the pulleys.

Having determined the exact position of the drum in relation to the shaft, it must be set off with great accuracy, either by measurement alone or with the aid of the theodolite.

The method of procedure is to get the exact centre of the shaft, or centre of barring on either side of the pit, by means of two cords, $a\,a^1$ and $b\,b^1$ (fig. 206), stretched across the pit at right angles to each other. From the centre of these two cords take another cord $o\,c$, equal in length to the distance the drum has to be placed from the centre of the shaft, and drive a stake at c. If the cord is in a straight line, the point c will be the centre line for the drum, but it should be tested by two side cords, $a\,c$ and $a^1\,c$, the exact lengths

of which can be calculated, since $a c^2 = o c^2 + o a^2$, and likewise $a^1 c^2 = o c^2 + a^1 o^2$. Both these cords should then be taken, and if the

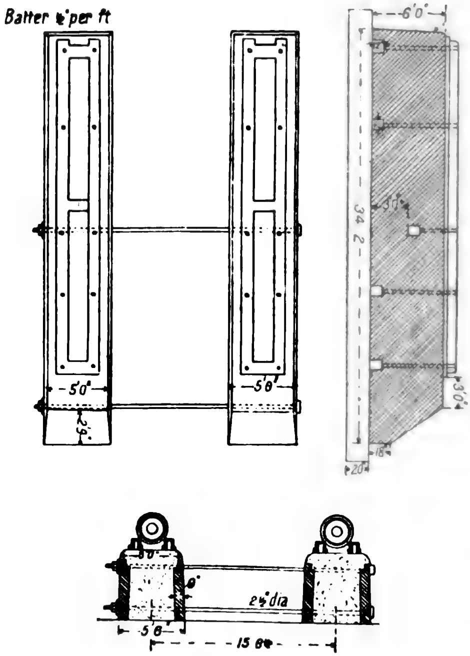


Figs. 207, 208, 209.

point c has been properly fixed, the ends of $a c$ and $a^1 c$ will also coincide with this point and give the centre line of the drum.

Engine Seats.—Winding engines are secured in their position by seats of wood, brick, or concrete, or a combination of the two latter. For small single cylinder engines wood seats may be used, as they are easily put into position and are cheap at first cost; but they have the disadvantage of being easily set on fire and are not so stable as brick or concrete. The wood may be either pitch pine or oak logs—generally the former is used—the principal beams being 18 in. to 24 in. square and the others 12 in. to 16 in. square. For an engine with cylinder 18 in. diameter, sixteen or eighteen beams would be required, and, as pitch pine costs 1s. 6d. per cubic foot, the cost of the engine seat would be from £20 to £30. Figs 207–209 show the arrangement of a wooden seat.

Brick or concrete seats are, however, preferable to those made of wood, as they give the engines a firmer foundation and a more solid bed, and are not susceptible to fire. Figs. 210–212 show the construction of such a seat composed of brick and concrete for a pair of horizontal winding engines, with cylinders 25 in. diameter. To fix the binding bolts, wood rhones 4 in. square are generally built in at the exact position for each bolt, and a set of 'pigeon-holes' left along the foot of the seat to fix the washers and cotter on the binding bolt. Brickwork for engine seats costs about 15s. per cubic yard, including labour, etc.; cement alone 18s. per cubic yard; and concrete alone, 18s. to 12s. 6d. per cubic yard.



FIGS. 210–212.—Plan, end and side elevation of brick and concrete seat.

Winding Ropes.—The different forms of winding ropes used are: (1) flat, (2) circular, (3) tapered; and the materials used in their construction are: (*a*) hemp or other fibres, (*b*) iron, (*c*) steel. Hemp ropes may be conveniently used for shallow pits and light loads, because of the facility with which they can be made to coil round small drums. They are also much used on winches for other colliery

work. The great objection to their use under other conditions is their weight; the weight of hemp rope for a breaking load of 20 tons would be about 20 lbs. per fathom, while the weight of a steel rope for the same breaking load (20 tons) would only be about 6 lbs. per fathom. On the Continent of Europe ropes made of Manilla and of aloe fibres are greatly in favour, even for very heavy loads and deep pits, and seem to be preferred to steel ropes, and they are said to give good results both as to wear and safety.

Iron Ropes are still a good deal used, and are recommended by some as superior to steel, both as regards wear and in affording better indications before breaking, besides being more pliable. But with the different qualities of steel now in use, these advantages over steel ropes no longer hold good, as varieties of the latter can now be had suitable for work under almost any circumstances.

There are four qualities of steel wire used for making winding ropes, viz. :—

				per sq. in. sectional area.
Extra plough steel	with	breaking	strain	of 110 to 120 tons.
Mild	,,	,,	,,	95 to 100
Best patent	,,	,,	,,	80 to 85
Bessemer	,,	,,	,,	40 to 45

For shallow pits where the load is light, it is found that Bessemer steel ropes are the most economical, because the first cost is considerably lower than that for those made from higher qualities of steel.

Flat Steel Ropes.—This kind of rope is not much used for winding, nor is it to be recommended for such purposes. It is more difficult to get a perfect flat rope than a circular one; the latter throughout being made by machinery, whereas the stitching of the flat rope is done by hand. The disadvantages of flat ropes are :—

Greater first cost, such ropes being 30 per cent. to 50 per cent. more costly than circular ones.
Very much shorter life.
Greater liability to failure.

Against this, the only advantage is the greater diameter of the drum of the descending rope which assists to lift the load at the beginning of the wind.

Circular Ropes.—These ropes are most largely used and are made of from four to seven strands, each strand consisting of five to thirty-seven wires, and for some purposes even more. Haulage ropes are preferably made of six strands, containing seven wires each. The strands are usually laid round a hemp core, made of long fibre Russian hemp, or where clips are used, as in haulage, of Manilla hemp, which has a harder fibre and is less liable to deteriorate. This hemp core should be carefully treated with linseed oil or other preservative, to prevent wasting from internal friction.

Wire of a soft quality steel is preferable for haulage, especially

where clips are used, and because it bends round quick curves with ease, and winds round small pulleys without injury. The essential features in a good winding rope are flexibility and strength, and it is desirable to obtain these qualities with the least possible weight. The weight of a winding rope is a very important matter; the *dead* weight to be lifted by the engine should be as little as possible. Another point to be considered is that the strength of the rope is in some degree dependent on its own weight, as the weight of the portion suspended in the shaft must be subtracted from that of the safe load.

Life or Durability of Ropes.—This will in a great measure depend on the treatment they receive and the work performed daily. Mr W. E. Hipkins states that the life of a rope will depend on the following points:—

> The quality and temper of the wire, having regard to the stresses the rope has to bear, and the conditions under which it has to work.
> Its construction as regards number of wires, strands, and nature of core.
> The ratio of the lay of its wires to that of its strands, and their proportions to the diameter of the drum or pulley over which it works.
> The nature of the dressing with which it is lubricated, and the mode and frequency of its application.
> The number and angle of the turns it is required to make in working.

All ropes ought to be well tested at stated intervals, by taking a piece nearest the cage and applying tensile and torsion tests to each individual wire. The tensile test consists in fracturing the wire by direct stress. The torsion test means that the wire must stand a certain number of twists in a length of 8 in. without cracking. The bending test is sometimes used at collieries, and consists in fixing a single wire in a vice, and then bending it at right angles a given number of times to see whether the wire shows signs of failure.

Winding ropes should also be re-capped at least every six months, as this gives an opportunity of examining the inside wires, and also changes the lifting point of the rope on the pit-head pulleys. Ropes should also be well dressed or lubricated, the lubricant to be applied with a stiff brush. Wherever practicable, the rope should be allowed to run through a trough, having brushes filled with the lubricant fixed on either side. Ropes treated with a good lubricant last from 25 per cent. to 50 per cent. longer. The dressing should be applied at least once a week. A good lubricant is made from the following ingredients: tar, summer oil, mica and axle grease, in varying proportions to suit varying conditions. The tar and oil must be free from acids. This combination is said to thoroughly penetrate the wires and prevents rust and so fills the cable as to give it an appearance of solidity; it resists water successfully, and does not strip. It is stated to cost only about one-eighth as much as ordinary lubricants, and to give better results.

Care and Management of Ropes.—Wire ropes ought to be care-

fully stored, and should on no account be placed on the ground, but upon sound planks raised several inches from the soil, so that they may be kept free from damp; they should also be covered over with tarpaulin and inspected frequently, besides having a coating of some lubricant at regular intervals. Care should be taken in uncoiling wire ropes to prevent 'kinking'; they should, during the process, be placed on a reel or drum when being 'paid' out. During working, the greatest stress on a rope being at the moment of starting, every care should be taken to ensure perfect steadiness, as jerking is very bad for ropes. No rope should ever be changed from a larger to a smaller drum, but it will do no harm to change it from a smaller to a larger.

If the following precautions are taken and carefully carried out, few accidents will occur to winding ropes:—

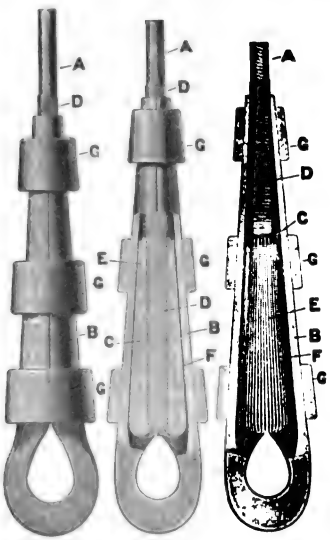
Choose a good quality of rope from a maker of good repute, and pay a fair price for it.
Make minute examinations of the rope at frequent intervals.
Protect the rope as far as possible from the action of the atmosphere and from water by frequently lubricating it.
Recap the rope and reverse it every six months.
Test portions at regular intervals.
Discard the rope after it has been in use a certain fixed time, even if it is apparently sound, so far as outside examination shows.

A careful record ought to be kept of all ropes, showing the length of time at work and the quantity of mineral raised by them, and also the speed at which they worked, as it is only by doing so that a fair comparison can be instituted between different ropes.

Rope Cappings or Attachments.—The proper capping of winding ropes is of great importance, for it is at such attachments that the rope wears quickest, and consequently where it will be most likely to give way. The cappings are fixed on the ropes in a variety of ways.

In the old method, which is still used to a considerable extent, the capping is formed by two semicircular sockets which nearly surround the rope, thickened out at the bottom end and formed into a link for attaching to the cage chains (fig. 219). The rope is fixed in this cap by rivets which pass through the capping and rope. To secure the rope properly, a part of the end is taken and the wire strands frayed out and bent back on themselves, the part of the rope to which this is done being firmly wound with tarred cord and tapered upward, to suit the shape of the capping or socket. When this is finished the socket is fitted on, and holes are carefully made through the rope with a marlinspike, to coincide with the rivet holes in the capping; as each hole is made, a rivet is driven through and well hammered when in position. This method gives fairly good results if the riveting is done carefully, but there is always the possibility of damaging the wires when the rivet holes are being made.

Another method is to use a socket with hoops, as in figs. 213–215. The rope is treated as already described, and drawn into the socket, and the rings then hammered firmly down into position. A third method is to use a solid conical socket or capping, which requires neither rings nor rivets (see fig. 216). The capping is made with a conical opening, and through this opening, and for some feet beyond, the end of the rope is drawn. The strands are now opened up as before, and laid back over themselves, some of the wires being cut off, and the rest carefully wound with copper wire until the end of the rope itself assumes a conical form; it is now drawn into the socket and is ready for use. Except under very exceptional circumstances, it will be impossible to draw the thick end



A = wire rope.

B = socket or capping.

C = hollow conical plug.

D = wire lapping on rope.

E = ends of wire of the rope turned back over the cone.

F = wire binding.

G = loose rings.

FIGS. 213–215.

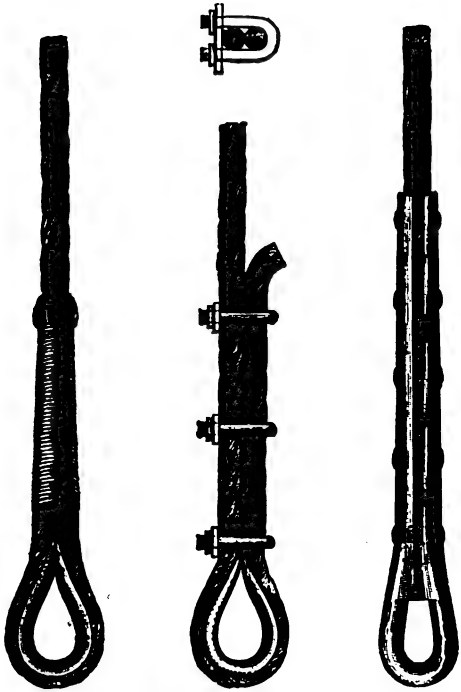
FIG. 216.

of the rope through the small end of the socket unless the capping were to split, which rarely happens. For additional security where heavy loads require to be raised, a collar is shrunk on. Figs. 218, 219 show the method of attaching flat and circular ropes respectively with capels and clamps.

Strength of Ropes.—The strength of ropes naturally varies according to the quality of the material of which they are constructed. In winding-ropes a large margin of strength should be allowed, and the gross load, including the weight of the rope between the pit-head pulley and the cage at the commencement of the lift, should never—except in rare cases—exceed one-tenth of the breaking strain.

The following formulæ are often used for finding the size of ropes

for a given load, or to calculate the breaking load for a given size of



FIGS. 217, 218, 219.

rope. These formulæ give only approximate results, and are not strictly correct in every case.

SIZE OF ROPES.

Let W = breaking load in tons = safe working load × factor of safety (M).
C = circumference of rope in inches.
M = factor of safety (10 for winding and 6 for haulage ropes).
w = safe working load in tons = coal + cage + tubs.

$$A\begin{cases}(1)\ \text{Then } W = C^2 \times \cdot 25 \therefore C = \sqrt{\dfrac{W}{\cdot 25}} \text{ or } C = \sqrt{\dfrac{w \times 10}{\cdot 25}} \text{ for hemp ropes.}\\[2ex] (2)\ \qquad W = C^2 \times 1\cdot 50 \therefore C = \sqrt{\dfrac{W}{1\cdot 5}} \text{ or } C = \sqrt{\dfrac{w \times 10}{1\cdot 5}} \text{ for iron ropes.}\\[2ex] (3)\ \qquad W = C^2 \times 3\cdot 00 \therefore C = \sqrt{\dfrac{W}{3}} \text{ or } C = \sqrt{\dfrac{w \times 10}{3}} \text{ for patent steel ropes.}\\[2ex] (4)\ \qquad W = C^2 \times 4\cdot 00 \therefore C = \sqrt{\dfrac{W}{4}} \text{ or } C = \sqrt{\dfrac{w \times 10}{4}} \text{ for plough-steel ropes.}\end{cases}$$

These formulæ do not allow for the weight of rope hanging in the shaft, and to correct this, a second formula may be employed.

Let L = lead of full cage in tons.
d = depth of pit in fathoms.
M = factor of safety.
C = circumference of rope in inches.

$$B\begin{cases}(1)\ \text{Then } C=\sqrt{\dfrac{L}{\dfrac{1\cdot5}{M}-\dfrac{d}{1\cdot1\times2240}}} \quad . \quad . \quad . \quad . \quad . \quad \text{for iron ropes.}\\[2ex](2)\qquad C=\sqrt{\dfrac{L}{\dfrac{3}{M}-\dfrac{d}{1\cdot1\times2240}}} \quad . \quad . \quad . \quad . \quad . \quad \text{for steel ropes.}\end{cases}$$

Examples.—Find the circumference of (*a*) a hemp, (*b*) iron, (*c*) plough-steel rope for a safe working load of 4·5 tons.

(*a*) $C=\sqrt{\dfrac{4\cdot5\times10}{\cdot25}}=\sqrt{180}=13\cdot4$ inches for a hemp rope.

(*b*) $C=\sqrt{\dfrac{4\cdot5\times10}{1\cdot5}}=\sqrt{30}=5\cdot47$,, ,, an iron rope.

(*c*) $C=\sqrt{\dfrac{4\cdot5\times10}{4}}=\sqrt{11\cdot25}=3\cdot35$,, ,, a plough-steel rope.

Or, by second formulæ, supposing depth of pit was 150 fms. :—

Then $C=\sqrt{\dfrac{4\cdot5}{\dfrac{4}{10}-\dfrac{150}{1\cdot1\times2240}}}=\sqrt{\dfrac{4\cdot5}{\cdot4-\cdot06}}=\sqrt{\dfrac{4\cdot5}{\cdot34}}=3\cdot63$ in. for a plough-steel rope.

WEIGHT OF ROPES.

C { Let w = weight of rope in lbs. per fathom. (1) Then $w=c^2\times\cdot25$ for hemp ropes.
c = circumference of rope in inches. (2) $w=c^2\times\cdot9$ for steel ropes.

The weight of rope in above calculations would be:

$w=(13\cdot4)^2\times\cdot25=44\cdot89$ lbs. per fathom for hemp rope.
and $w=(3\cdot63)^2\times\cdot9=11\cdot90$,, ,, steel rope.

STRENGTH OF CAGE CHAINS.

D { Let w = safe working load in tons.
D = diameter of iron in eighths of an in. ($\frac{1}{8}$-ths). Then $w=\dfrac{D^2}{2\cdot5\times M}$.
M = factor of safety (10 for cage chains). $\therefore D=\sqrt{w\times M\times2\cdot5}$.

Example.—What size or diameter of iron should be used in cage chains for above calculations?

$$D=\sqrt{4\cdot5\times10\times2\cdot5}$$
$$=10\cdot6 \text{ eighths or } \frac{10}{8}=1\tfrac{1}{4} \text{ in. diameter of iron.}$$

To find the weight of chains,

Let W = weight in lbs. per fathom.
D = diameter in sixteenths of an inch.

Then $W = D^2 \times \cdot 21$; for above size of chain $W = (20)^2 \times \cdot 21$.
$= 84$ lbs. per fathom.

The weight varies with the length of link, but for ordinary makes the above gives the average weight.

Counterbalancing.—The load of a winding engine is often a very varying quantity, particularly in deep shafts, and the power of the engine cannot under such circumstances be utilised to the best advantage, hence the necessity of some compensating arrangement to equalise the load during the 'wind.' Balancing the load can be effected by different methods, such as by—

(1) The chain and staple arrangement.
(2) By the tail rope method.
(3) By using conical or spiral drums.
(4) By Koepe's system of winding.

Chain and Staple Arrangement.—In this method of counterbalancing, a staple pit is required in addition to the winding shaft (fig. 220). The depth of this staple is such that when the cage is at the beginning of the wind, the heavy chain which is attached to the drum shaft, and which is used as a balance, will be hanging at the top of the staple; its weight at this position will assist the winding-engine when most required, *i.e.* at the start of the lift, and when the cages are at mid shaft the whole of the chain will have accumulated at the bottom of the staple.

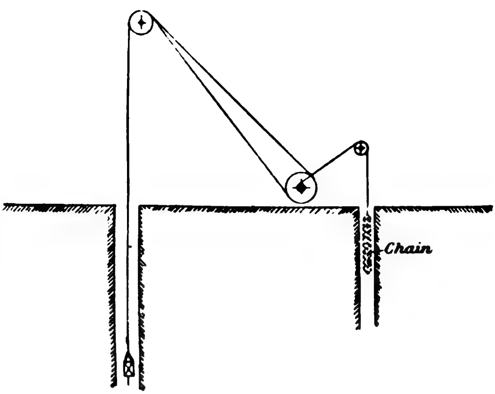


FIG. 220.—Chain and staple balance.

During the latter half of the wind, when often the load is a negative quantity, the chain will again be raised to the top of the staple ready for another wind. Thus during the first half of the wind, and when the load is greatest, the engine is assisted; in the latter half, when the load is a diminishing quantity, the engine is retarded and brought more easily to a stop. The disadvantage of this method is, that it often requires a staple pit 40 to 50 yds. deep, involving considerable expense.

Tail Rope Method.—In this system of balancing, a 'tail rope,' of the same weight as the winding ropes, is attached to the bottom of each cage, and passes round a beam, placed across the shaft, and below the level of the pit-bottom (see fig. 221). A pulley working in a sliding frame was at first used in this method instead of a beam, but it does not work so well, as the rope is apt to get off the pulley, or to pull it out of position. When the pit is deep no pulley or beam is required, the weight of the rope causing it to form a natural loop of itself. This system of counterbalancing has been found to give good results, but it is most suitable for shafts that are comparatively free from cross buntons, pump rods, pipes, and other apparatus, although the writer has seen it successfully applied in rectangular shafts, where cross buntons are necessary. The tail rope should be attached to a bar of iron laid across the bottom of the cage, the strength of the bar being less than that of the rope, so that in case of accident the latter may give way rather than the rope.

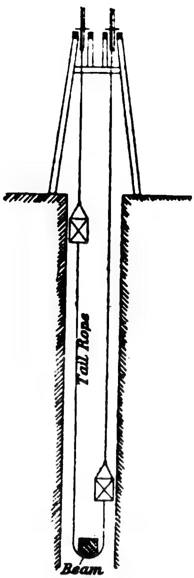


FIG. 221.—Tail rope method.

Conical or Spiral Drums.—This method of counterbalancing would be one of the best that could be adopted, but to obtain perfect balance, and at the same time to get a satisfactory degree of wear out of the ropes, the drums would often require to be 25 or 30 ft. in diameter, which would make them very heavy and expensive. Hence such drums are not much used, and where they were employed, they have been discarded owing to accidents taking place by the rope slipping on the drum. To prevent slipping, a spiral groove was sometimes turned on the drum. In many cases the spiral is put on the drum by riveting an angle iron on shell plates or an openwork frame, the angle iron forming a continuous spiral from end to end of the sloping

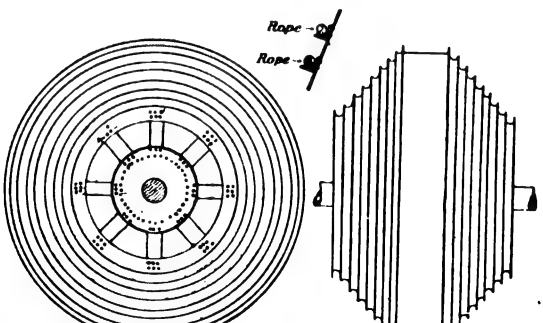


FIG. 222.—Conical or spiral drums.

portion of the drum (fig. 222). By this method it is almost impossible for the rope to slip. The drum is usually so arranged that several coils of the rope wind round the flat centre part of the drum. This enables the latter to be made smaller at the large diameter than would otherwise be the case.

To find the ratio between the large and small diameters of such drums the following rule may be used : —

Let a = full cage at pit-bottom.	d = empty cage at bottom.
b = empty cage at pit-top.	x = diameter of small drum in ft.
c = loaded cage at pit-top.	y = ,, large ,,

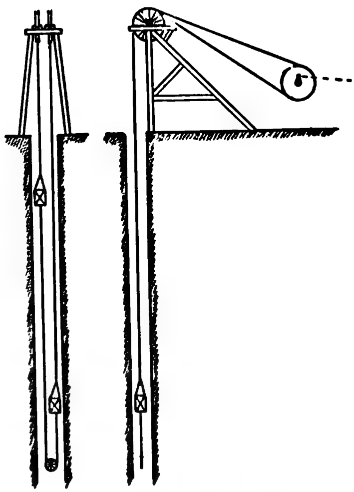
$$\text{Then}\left\{(a\times x)-(b\times y)\right\}=\left\{(c\times y)-(d\times x)\right\}\quad\therefore\ x=\frac{y(c+b)}{a\times d}\ \text{or}\ y=\frac{x(a+d)}{c\times b}.$$

Example.—Suppose full weight of cage at pit-bottom is 4 tons, full cage at top is 3 tons, empty cage at bottom 4 tons, and empty cage at top $1\frac{1}{2}$ tons; let small diameter of drum be 12 ft.; find what diameter large drum would require to be under above conditions.

$$\text{Here } a=4,\ b=1{\cdot}5,\ c=3,\ d=4,\ \text{and } x=12.\quad\therefore\ y=\frac{12(4+4)}{3+1{\cdot}5}=21{\cdot}33 \text{ ft.}$$

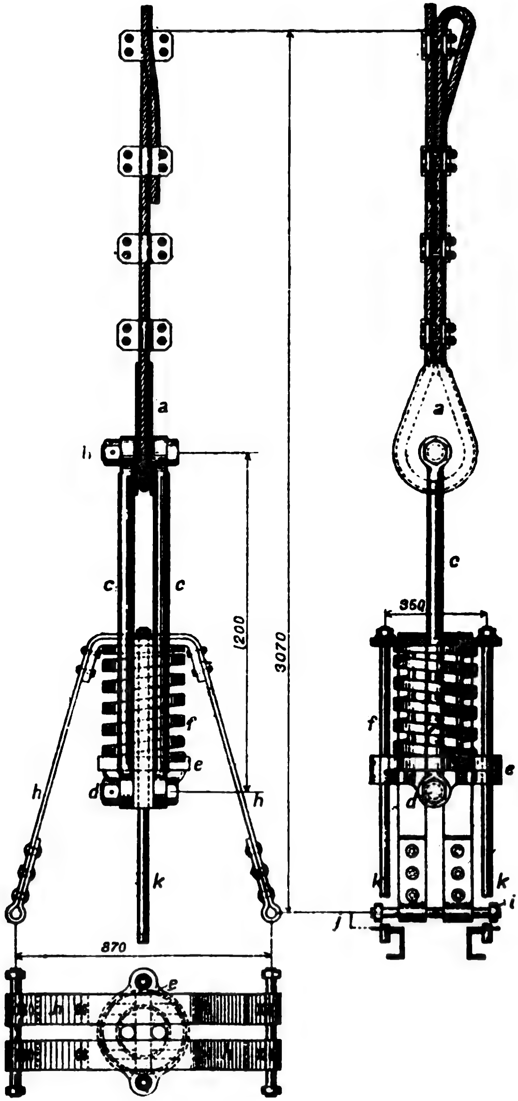
$\therefore$ y, the diameter of the large drum, would require to be $21\frac{1}{2}$ ft.

Koepé's System of Winding.—This system of winding was invented to secure a properly balanced load. Instead of a drum, as in ordinary winding arrangements, a large pulley is used, and only a single winding rope is required. The same rope extends all the way from one cage up over a pulley on the head-gear, round the engine pulley, back over the second pulley on the head-gear, down to the other cage, then to the bottom of the shaft, where it forms a loop, and lastly up to the cage from which it started. The friction between the rope and the engine pulley is sufficient to raise the useful load, which represents all the work required to be done by the engine, since the rope is balanced in every position of the cages.



FIGS. 223, 224.—Koepé system of winding.

This system has never had any extended application for winding in Britain. At the few places where it was adopted, it has since been discarded for the ordinary arrangement. In working, it was found that the winding pulley did not give sufficient gripping power, and the rope was apt to slip, and



FIGS. 225, 226.—Spring attachment for winding ropes.

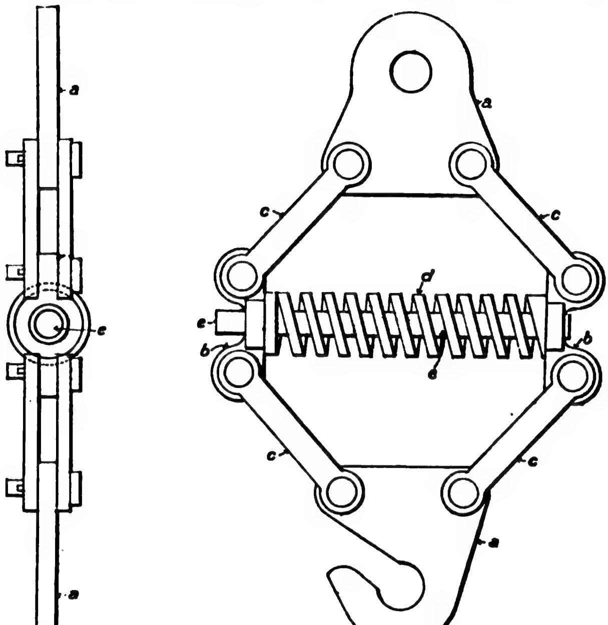
allow the cages to run back. Another danger was that if the rope broke, both cages would fall to the bottom. To overcome these difficulties, a modification of this system has been adopted, in which two additional pulleys are used below those of the head-gear, and at right angles to them, over each of which a rope extends from one cage to the other. These safety ropes are meant to prevent the cages from falling away in the event of the main winding rope breaking. A balance rope is also used, which is attached to the bottoms of the cages and passes round a beam in the pit-bottom.

Reducing the Strain on Winding Ropes.—As the greatest strain on a winding rope is at the moment of starting, various appliances have been introduced to reduce this as much as possible. One of the first methods was to put pieces of india-rubber below the 'mingles,' or plummer blocks, to give some spring to the pulley when the load is lifted. But rubber is a bad substance for this purpose, as it possesses a very small amount of elasticity, and soon loses the little it has, especially when exposed to the weather. Placing the pedestal on springs, like an ordinary waggon spring reversed, has been tried with much success for reducing the strain on the rope at starting to wind.

Spring Attachment.—At the Gerhard Colliery, near Saarbrücken, in Germany, a spiral arrangement is used for attaching the rope to the cage. The former is placed round a sheave *a* (figs. 225, 226), and the loose end is fixed to the rope by means of four clips. A bolt *b*, $2\frac{1}{2}$ in. diameter, runs through the sheave, to which two round wrought iron rods *c* of a diameter of 2 in. are attached at their lower ends. These rods are connected at the bottom by another strong bolt *d*. Upon this last bolt rests a plate of cast steel *e*, which receives the powerful spiral spring *f*. The upper end of the spring acts upon the wrought-iron plate *g*, above which two hoops *h* are screwed. The ends of these hoops are formed into eyes, which by means of the bolt *i*, $1\frac{5}{8}$ in. diameter, are connected with the links riveted on the cages. The round rods *k* serve as guides to the plate *e*. When the cage is raised, the spiral spring presses against the iron plate under the hoops, and the cage is lifted gently. When the rope above the cage is loose, the rods *c c* descend perpendicularly on to the plate, and thus all jerking of the rope is avoided.

Spring Coupling.—Another apparatus used for the same purpose is that known as the 'spring coupling.' Figs. 227, 228 show this apparatus. It consists of two plates *a a*, working on pivots, and two end plates *b b*, connected by links *c c*, which are pivoted to both of these plates. The side plates are held apart by a spring *d*, through the centre of which passes a loose rod *e*, one end of which is fixed rigidly. When a sudden strain is put on the end plates *a a*, the side plates *b b* approach each other and compress the spring. As the strain increases, the resistance of the spring to compression also becomes greater and the compressing power of the links becomes less, so that a condition of equilibrium is attained.

Cage Guides or Conductors.—*Wood Guides.*—The cage is directed in the shaft by guides, which may be made by either (*a*) wood, (*b*) iron or steel, (*c*) wire rope or circular rods of iron. Wood guides are usually made of pitch pine, the size depending on the loads raised. For cages holding one tub the guides would be 4 in. × 3 in. or 4 in. × 4 in., and for double cages with two tubs, 5 in. × 4 in. or 5 in. × 5 in. The guides are cut into lengths of 12 to 30 ft., 18 and 24 ft. being lengths commonly used. They are fixed to the cross

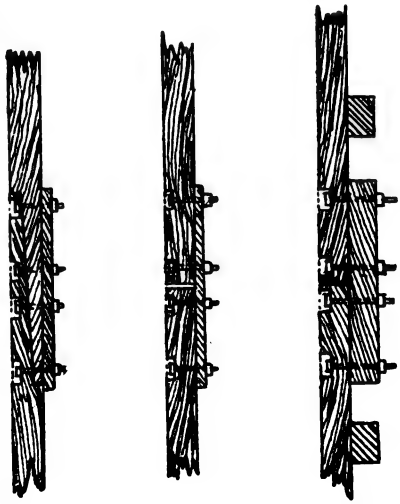


FIGS. 227, 228.—Spring coupling.

buntons in the shaft, and joined together as shown in figs. 229, 230, 231. Fig. 231 shows a common method of joining them by an ordinary 'butt' joint, with a piece of wood 4 in. × 4 in. to stiffen them, fixed at the joint by means of bolts having their heads counter-sunk in the face of the guide so as not to catch the shoe, attached to the cage, in passing; or very often an iron plate about $\frac{5}{8}$ in. × $3\frac{1}{2}$ in. is used with an ordinary butt joint as before (see fig. 230). A 'scarf' joint is also used and an iron fixed as before, with bolts having their heads counter-sunk (see fig. 229). Wood guides are most suitable for rectangular shafts in which cross buntons are necessary for their construction; they are easily fixed, cheap at

first cost (about 1s. 6d. per cubic foot), but on becoming worn require frequent repairs and are very liable to cause accidents.

Iron and Steel Guides.—These guides are now very largely used. They are made somewhat in the style of an ordinary rail for surface use, and weigh from 40 to 60 lbs. per yard Great care is required in fixing these guides to fit them to the proper gauge, and to have all joints even and perfectly vertical, because if they are not well-fitted to begin with, they give a great deal of trouble.

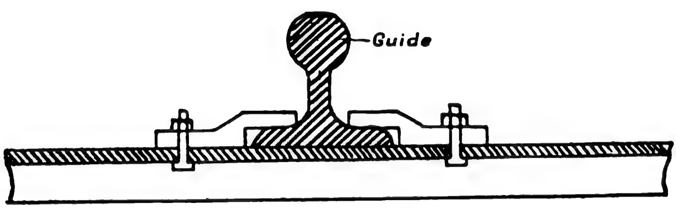


Figs. 229, 230, 231.—Cage guides.

Fig. 232 shows the ordinary method of fixing them; each 'chair' should carry the weight of the guide above it, as they should not rest on one another, but a small space should be left at each joint to allow for contraction and expansion. The space for admitting the guide into the chair is made a little smaller for the top side, so that a small portion requires to be taken out of the guide to admit it, the projecting parts acting as a support for it.

These rail guides should be made of steel, as iron, having very much less elasticity, causes a greater degree of vibration in the cage, which is a matter of great importance in rapid winding.

Briart's Method of Fixing Guides.—In this method of fixing steel

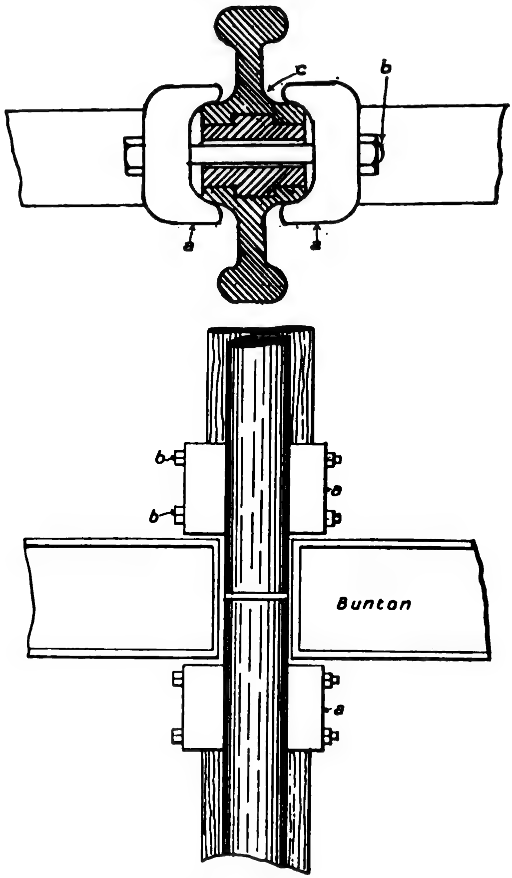


Figs. 232.—Iron guides.

guides, which is so common on the Continent, a single series of H girders, 9 ft. 10 in. apart, divide the shaft.

Each guide is 19·66 ft. long, which allows a slight play between the joints. Previous to being fixed, the buntons are carefully notched to receive the foot of the rail to a depth of 0·39 in. and to a width

of 4·33 in. Figs. 233, 234 show the method of fixing the guides. Two steel glands *a a* secure the rail *c* to the buntons, one above and another below, and are made to grip the guide firmly by a pair of bolts *b b* passing through them. To prevent any movement, or the rails from getting twisted by the pressure of the glands, a block of cast

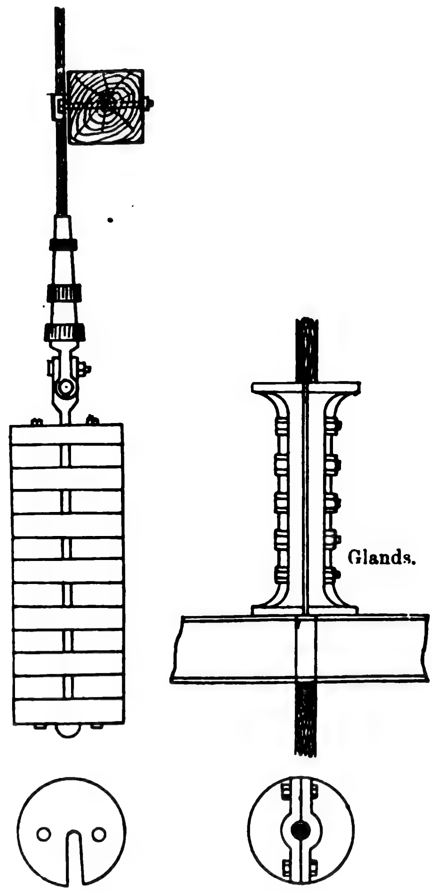


Figs. 233, 234.—Briart's method.

iron, through which the bolts pass, is placed between the rails, and is furnished with a slight projection, which fits into a corresponding groove in the flange of the guide. This arrangement has been found to act very effectively. In passing through tubbing, the buntons are carried in boxes or shoes bolted to the internal flanges of the tubbing, the girders being wedged in position with wood keys.

Iron or steel guides are suitable for either circular or rectangular shafts, and always require buntons for fixing them; they are much more durable than wood guides, but more expensive at first cost and difficult to repair. When they get unevenly worn they can no longer be repaired, and there is also a tendency for the circular head of the guide to get worn off if the cage shoe grips it too tightly. If properly fitted at first they will often last for years without giving any trouble, and require little repair.

Rope or Rod Guides.—These conductors are very extensively used in circular shafts where no buntons are required in their construction, and there can be no doubt as to their suitability. It must, however, be remembered that there should be at least 15 in. to 18 in. clearance between the corners of the cage and the walling to ensure safety in winding. Cold drawn steel rods twisted together are often used instead of the ordinary rope guides; they give greatly increased strength, and wear much better. They are made up of seven to fifteen rods, with a total circumference of $2\frac{1}{4}$ to $4\frac{1}{4}$ in., and weigh from $8\frac{1}{2}$ to 25 lbs. per fathom. The size used will depend on the load and strength required. The conductors are fastened to a strong beam at the pit-head by means of five or six glands to keep them from slipping (figs. 235–238), or they may be fastened by a capping and strong eyebolts fastened with a nut and washer.



Figs. 235, 236, 237, 238.—Rope guides.

At the pit-bottom they are kept tight by attaching weights to them (fig. 235). This is preferable to fixing them rigidly, as it allows

for expansion and contraction, which is often of considerable amount, the weight used being about 1 ton for 250 yards of conductor, although the exact amount required can only be ascertained by experiment in each individual case. The weight should not be in the form of a single solid block, but in the form of segments or 'cheese weights,' and used in much the same way as weights are used on a 'dead-weight' safety valve. Some prefer to fix the guides at the pit-bottom and pass them over pulleys at the surface, attaching weights to the loose end. At some collieries they are fixed by a strong spring, to allow of the necessary expansion and contraction.

The number of conductors used will depend on the load and upon the speed of winding, but in ordinary cases two guides are often used one on each side of the cage, while for heavy loads and quick winding it is best to have at least four guides, one at each corner of the cage.

To prevent the cages from catching each other in passing, two unconnected conductors should be suspended between them; the space between the cages may then be as little as 6 in., but 12 to 15 in. is a better allowance.

The advantages of using rope guides are:—

(1) The first cost is cheap.
(2) They are easily fixed and require few repairs and little attention, except oiling regularly.
(3) They last much longer than wood or iron guides.
(4) No buntons are required for fixing, and they occupy but little space in the shaft.
(5) They are more flexible, and contract and expand more readily than rigid wooden or rail conductors.

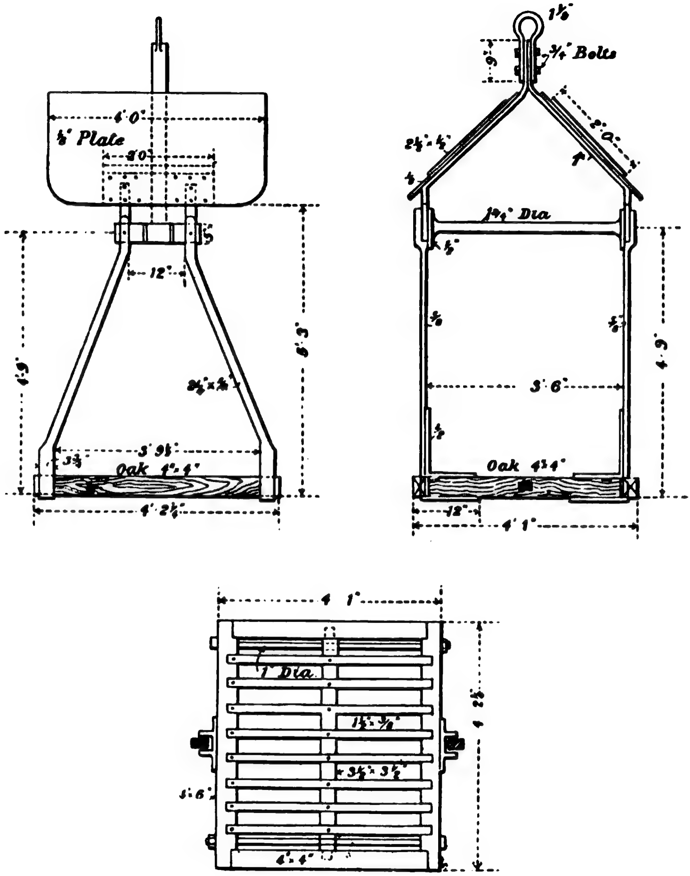
Cages.—These are made of different sizes according to the output required, the size of shaft, and that of the tubs used. The material employed in their construction is either wrought iron or steel, the bottom being often constructed of oak, but for large cages it is best to dispense with wood altogether, and to make them entirely of steel or wrought iron, as by this means their weight will be much reduced, which is an important point to keep in view. A good cage made of steel throughout should not weigh more than about two-thirds the weight of the coal raised, while a wrought-iron one should weigh about three-fourths of weight of coal raised per wind.

A cage should be so constructed as to allow the greatest carrying capacity with the least possible weight; the form selected should be such that the tubs can be readily placed in it and easily removed, while the whole construction should run easily in the shaft.

Cages cost from £30 to £35 per ton if made of wrought iron, and from £40 to £45 per ton if constructed of steel. Before cages are used they should have two coats of good paint, which will help to preserve them from corrosion in wet shafts, and particularly in pits where the water contains acid. Figs. 239, 240, 241 show the con-

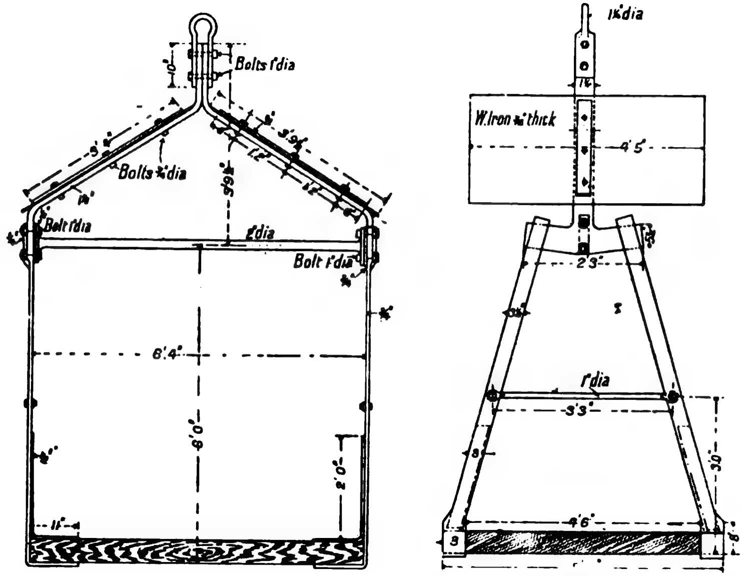
struction of cages for holding a single tub, this form shown being frequently used in Scotland, as they are easily and cheaply made.

Figs. 242, 243 show a form of single-decked cage for two tubs placed abreast, a type much used in rectangular shafts.

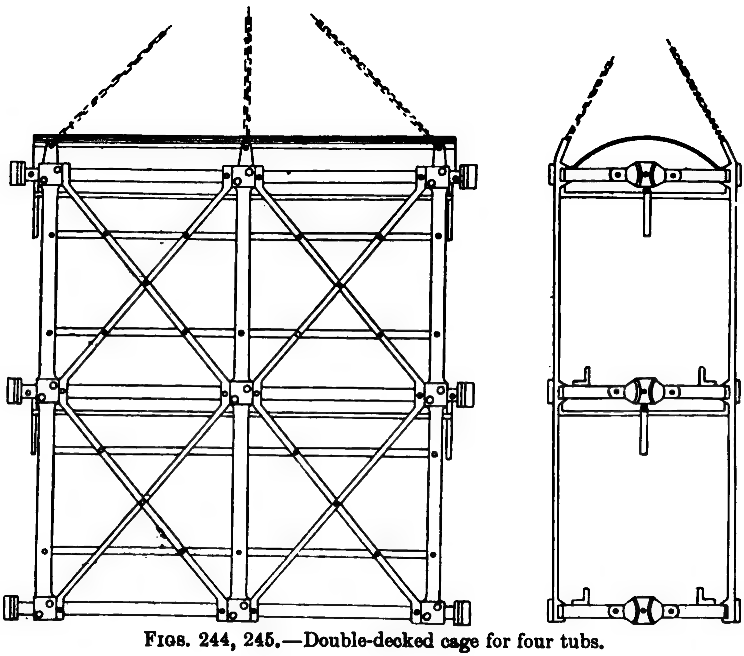


Figs. 239, 240, 241.—Single cage.

Figs. 244–247 show two types of double-decked cages, in one of which the tubs are placed abreast, and in the other, which is usually more suitable in circular shafts, end to end. Cages with the tubs placed abreast are generally more suitable for rectangular shafts, where the space is narrow compared with its length. Figs. 248, 249



Figs. 242, 243.—Single-decked cage for two tubs.

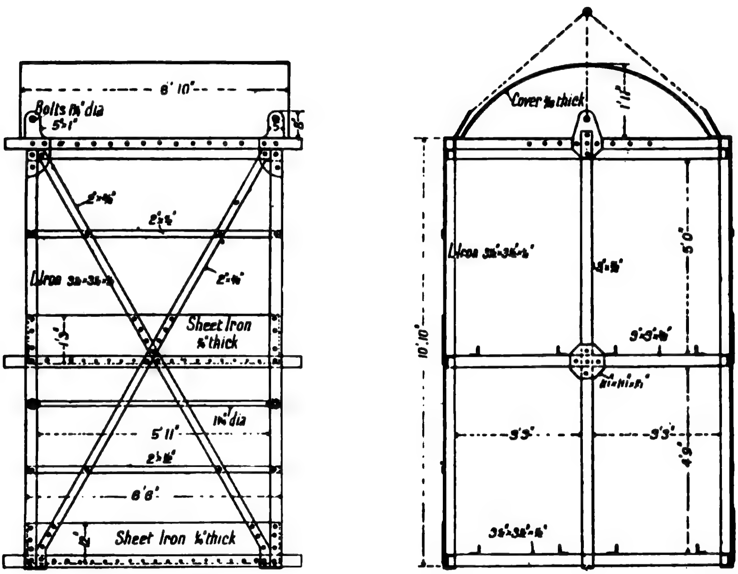


Figs. 244, 245.—Double-decked cage for four tubs.

show the detailed construction of a cage for drawing mineral on an inclined shaft.

Cage Speeds.—The speed of the cage in the shaft may be anything between 15 and 40 ft. per second. The following are average speeds: —depth, 50 to 100 fms., speed, 15 to 20 ft. per second; 100 to 150 fms., speed 20 to 25 ft. per second; 150 to 200 fms., 25 to 35 ft. per second; and for depths above 200 fms., from 35 to 40 ft. per second.

Pit-head Pulleys.—Pulleys used on pit-head frames are usually made either of cast or wrought iron, or a combination of both, the rim and bosses being cast and the spokes constructed of wrought iron. The shape of the groove varies with the shape of the rope

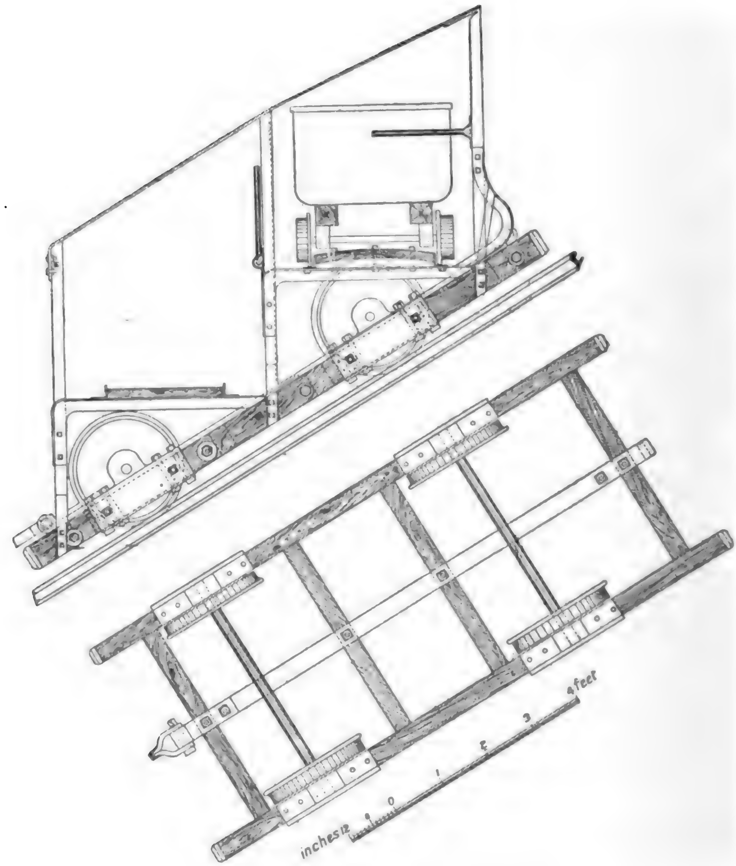


FIGS. 246, 247.—Double-decked cage for four tubs.

used. The size of pulleys will also vary between 6 and 20 ft. in diameter, according to the size of rope and drum used. Where wire ropes are used they should be as large as possible to avoid straining the rope by too sharp a bend, a common size being between 12 to 16 ft. in diameter. The diameter should, however, be in proportion to that of the rope used.

For a steel winding rope 1 in. diameter, a pulley not less than 10 ft. in diameter is required, and for ropes from $1\frac{1}{4}$ in. to $1\frac{3}{4}$ in., pulleys should be 15 to 20 ft. in diameter, while, as a rule, they ought to be about the same size as the winding drum.

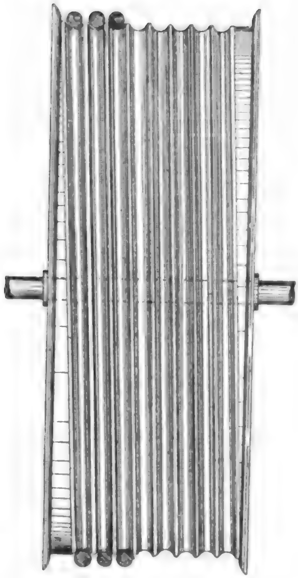
Drums.—Winding drums are either cylindrical, conical, or spiral in shape, the first-named being the most common, and being usually constructed of two cast-iron cheeks fitted or keyed on to a wrought iron shaft, and lagged between the two cheeks with strong wood deals for the rope to coil on. Sometimes they are wholly constructed of



Figs. 248, 249.—Cage for inclined shaft.

wrought iron or steel. Conical drums should have a spiral groove running round them, from the shorter to the longer diameter, in order to keep the rope from slipping. A spiral groove should also be used on an ordinary cylindrical drum to prevent excessive side friction between the coils of rope when winding, which will increase its life (see fig. 250). Drums ought to be made as light as possible so as to

minimise the inertia at starting. Good, strong, and at the same time light drums are best made of steel.



The size of drum used will greatly depend on the size of engine erected for winding and upon the size of the rope, but it is a mistake to use very large drums, as large diameters decrease the efficiency of the engine. In winding, speed requires to be got up quickly each wind, and with large drums this will not be so easily attained as with drums of smaller diameter.

The drum may be increased 1 ft. in diameter for each additional 2 lbs. weight in the rope per fathom.

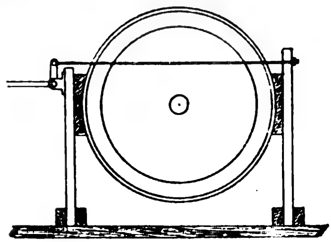


FIG. 250.—Drum.

FIG. 251.—Drum with block and brake.

Drum Brakes.—It is desirable that a good reliable and efficient brake should be attached to the drum of all powerful high-speed winding engines. The power should be applied as near to the centre of the drum as possible, and a large leverage, from 150 to 200 to 1, given. The ordinary block brake is very efficient, and is largely used in Scotland (fig. 251). In this brake two blocks of wood, generally elm, are brought into contact with an iron ring fixed either on the centre or side of the drum, and the power applied by

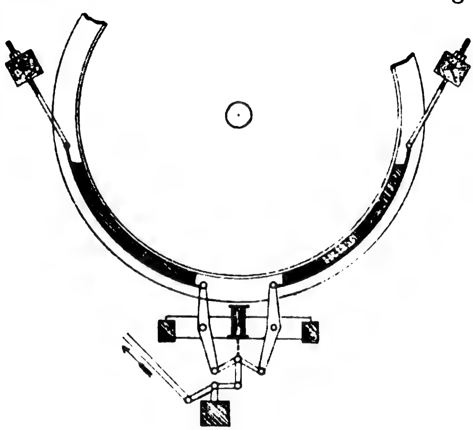


FIG. 252.—Brake device.

an arrangement of levers that can be worked either by hand or foot by the engineman. The type of brake shown in fig. 252 is much used in some districts of England. It is one which is arranged underneath the drum, and there is little friction when the engine is at work, as when released it immediately frees itself of all contact with the brake 'ring.'

This brake is applied by a toggle joint arrangement, and is arranged so that it can be worked either by hand or steam power. It should be made from well-seasoned blocks of elm or oak wood.

Another form of brake—known as Burns's brake—one that gives good results, is very powerful, and has also the advantage of being simple in construction, is shown in fig. 253. In this brake a single block of wood is fixed to a long lever and applied to the bottom of the drum, the leverage being in the ratio of 200 to 1. In the block holes are sometimes bored and filled with sand, which renders the brake much more effective.

Safety Hooks.—In no class of work about a colliery is there more liability to accident than in winding, and yet such accidents are happily rare, no doubt owing, in a large measure, to the careful

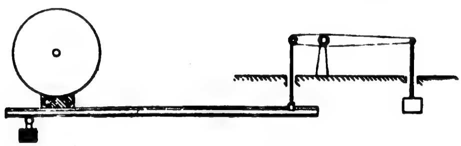
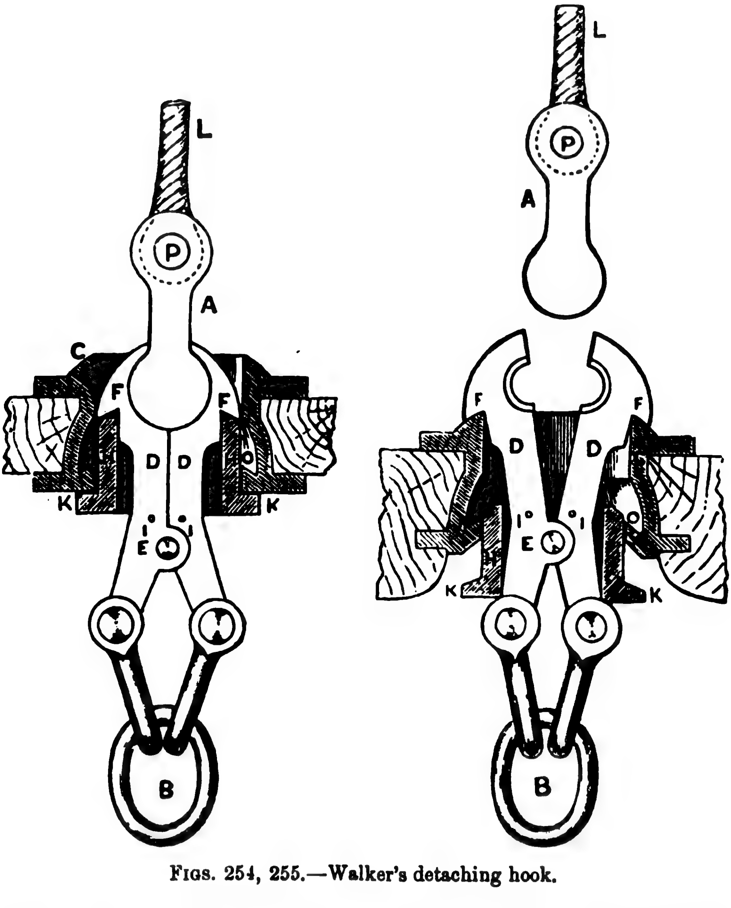


FIG. 253.—Burns's brake.

handling of the engines by those in charge. When it is considered, however, at what speed they have to be worked, and the number of times the cages have to be raised and lowered even in the course of one shift, it is obvious that an accident due to overwinding may easily occur even with the most careful engineman, as some portion of the engine may get beyond control, and prevent it from being stopped at the proper position. It was to obviate the effects of overwinding that safety hooks, which have been applied with much success, were invented. Many colliery owners do not, however, use these hooks or other safety appliances, because they are likely to get out of order and not act when required, and more confidence is placed in having good reliable men at the engines than in any mechanical contrivance. There have, doubtless, been cases where safety hooks have not fulfilled expectations, and where they have even broken when an overwind took place. But this may happen with any piece of machinery, particularly if it is not properly looked after and kept in good working order. If, on the other hand, the use of safety hooks renders men's lives safer in the event of an overwind, there is no good reason why they should not be adopted at all collieries.

There is a large variety of safety hooks before the public, but the principle upon which they act is practically the same in all. In some the rope is simply released when the cage is overwound, and in others the rope is released and the cage held fast simultaneously.

Walker's Detaching Hook.—This is one of the most efficient safety



FIGS. 254, 255.—Walker's detaching hook.

hooks in use. Its principle will be understood from figs. 254, 255. The hook consists of a pair of jaws D D working on a centre pin. These jaws are held together and made to retain the strong action bolt in the rope shackle A by means of the clamp K, which is kept in position by the copper pins I, and the outward pressure due to the weight of the load. In the event of an overwind, the jaws pass

freely into the ring C, which is a fixture, but the flanges K of the clamp H coming into contact with the ring C, as in fig. 254, is held stationary while the jaws are pulled through, with the result that the copper pins I are sheared off, and the hook jaws F F are forced open by their lower portions being drawn into the clamp, in which position they are firmly locked, as shown in fig. 255; the rope then passes over the pulley, and the load remains suspended.

This hook being made without side plates, is not liable to get fast, is simple in construction, and can be quickly and easily re-connected.

West's Hook.—This hook is also simple in arrangement, and is

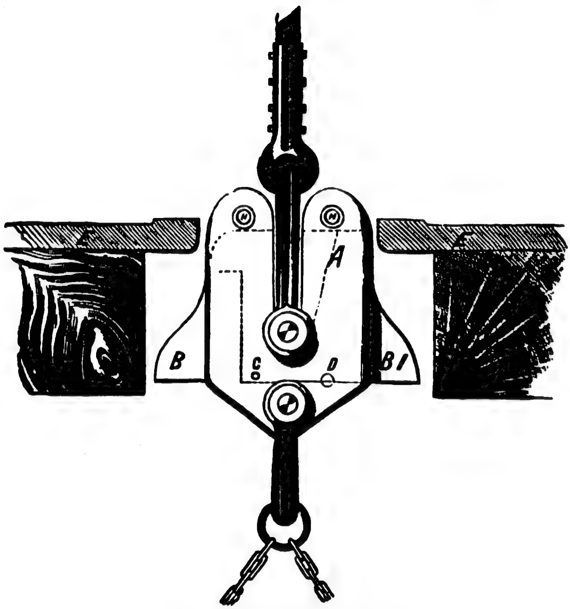


FIG. 256.—West's hook.

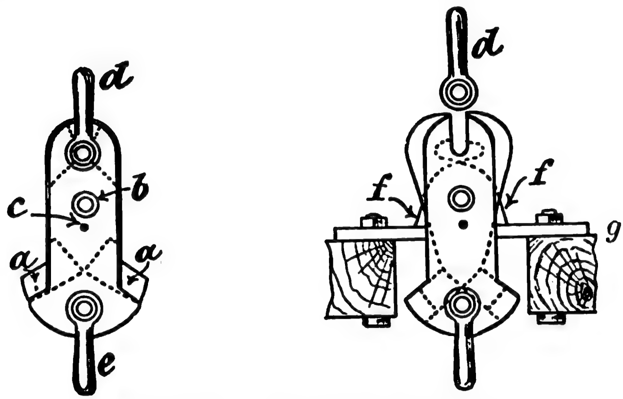
composed of the body A (see fig. 256) and two sliding catches B and B′, fitted with a copper releasing pin C and a locking bolt D.

When an overwind takes place, the wedge-shaped portions of the sides B B′ come in contact with a fixed plate E, and are forced into the outer steel box A, whilst the opposite ends are forced out as in the figure, allowing the shackle and pin to be liberated and held suspended on the plate E.

King and Humble's Hook.—This hook consists of two outer plates *a a* and two inner plates, all of which are pivoted upon a strong centre pin *b* (see figs. 257, 258). The winding rope is attached to the top shackle *d*, and the cage and chains to the bottom shackle *e*.

The wrought-iron catch-plate *g*, through the centre of which passes the winding rope, is securely fixed to the head frame immediately under the pulley wheel.

In the case of an overwind, the hook is partially drawn through the centre hole in the catch plate, until the bottom jaws of the inner plates of the hook come in contact with the underside of the catch-plate, when they are pressed inwards, shearing the copper pin *c*, causing, by the same action, the upper jaws to extend, thus releasing the rope, and, at the same moment, the hook locks upon the catch-plate. The latter is so constructed that there is just sufficient space between the lower jaws and the locking jaws for the catch-plate to insert itself, hence the hook cannot be sufficiently detached through the catch-plate to allow the locking jaws to get on the top side of the plate. As soon as the under jaws are forced out, the hook is



FIGS. 257, 258.—King and Humble's hook.

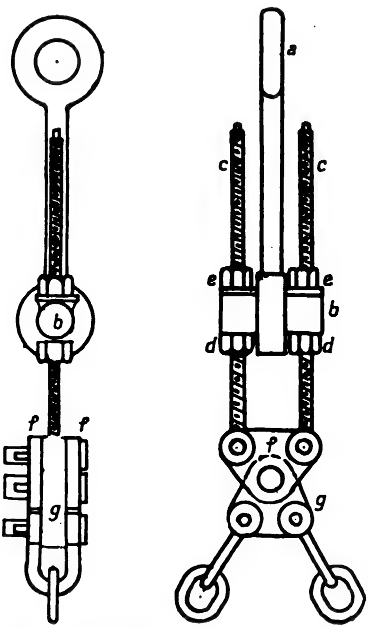
therefore locked on the upper side of the catch-plate. King and Humble's hook is also furnished, in case of an overwind, with an automatic lowering arrangement which consists of an elongated slot just above the centre pin. When an overwind occurs, the rope is brought back over the pulley to the hook, for which a spare shackle is provided. This is passed through the rope shackle and down over the hook to the lowering slot, whereupon the rope is slightly tightened, which causes the inner plates of the hook to close, and the hook with the cage attached can now be lowered on to the pit keps.

Safety Cages.— Safety hooks, such as those described above, are meant only to prevent accidents in cases of overwinding, and afford no security against accidents resulting from the rope breaking while the cage is running in the shaft. To guard against this, innumerable safety cages and appliances have been invented, although few of them have proved to be of any real value in practical working.

On the Continent, as in Germany, the use of safety cages is enforced by law ; in Britain, however, such appliances are not compulsory, and among colliery proprietors, at least in their present form, find but little favour. Most of them depend for their action on a grip or spring which ordinarily is not in contact with the guides, but which, in the event of the rope breaking, is released, and clutches them in order to prevent the cage from falling. While they may be of some use where winding is carried on at low speed, they are practically useless at most modern collieries where the speed is often very high. In such cases they often fail to act on an emergency, or allow the cage to fall back with such velocity that the guides are greatly damaged or even broken, and the cage is precipitated to the bottom of the shaft. If men are in it, the shock is likely to be so great as to either pitch them out or dash them against the top of the cage.

Quite recently an accident occurred in a mining district in Germany, with one of these protected cages, supplied with safety grips and a controlling lever worked from the cage itself.

Notwithstanding these precautions, and the fact that everything was in working order, the appliances proved useless, and gave way, with the result that the nine men in the cage were killed.



FIGS. 259, 260.—Adjusting screws for winding ropes.

The best preventive against such accidents occurring is to use only the best quality of winding ropes, to give them careful treatment, and to inspect them frequently.

Adjusting Screws.—In ordinary practice the length of winding ropes is adjusted by increasing or diminishing the spare coils on the drum and by refixing them or cutting a portion off and re-capping. By such methods it is, however, very difficult to secure exact adjustment without much labour and care. To obviate this, adjusting screws have been applied. Figs. 259, 260 show the construction of these screws.

They consist of a strong steel rod *a*, terminating at each end in an eye. The shackle at the lower end of the rope is attached to the upper eye. A round block *b*, with a hole in each end large enough

to admit of the easy passage through it of the screws *c c*, is placed in the lower eye. Screw *c*, $2\frac{1}{2}$ or 3 ft. long, with strong threads and an eye at their lower end, and provided with nuts *d*, screwed on to them, are passed through each hole in the block, and the nuts *e* are then screwed on to them from above. Each nut *e*, while resting on the block, supports its own screw. Two triangular plates *f f*, with a hole at each angle, are attached, by means of pins passing through two of their holes, to the eyes at the lower ends of the screws, which they then enclose between them. The third eye in each of these plates hangs vertically below the steel rod which supports the block. A third triangular plate *g*, with three holes, one at each angle, is inserted between the two first, and a pin is passed through one of these holes and through the unoccupied holes in the two plates above it. Two short pieces of chain are attached to the remaining holes in the lower triangular plate by means of shackles, the cage being attached to these chains. The ropes can be adjusted in a very short time, without much labour, by means of these screws.*

Cage Props or Keps.—'Keps' or 'props' are required at most collieries, as a rest for the cage and to keep it in position during the changing of the tubs. At some collieries no keps at all are used, the engineman maintaining the cage in position by applying the brake to the winding drum until the tubs are changed, by which means the ropes are said to last longer. It is also claimed that there is less liability to accident, owing to the absence of 'keps,' which require to be opened and shut every time the cage comes to the surface. Unless the cage, however, is brought to a dead level with the plates on the pit-head, the rope undergoes a good deal of jerking during the operation of changing tubs. The ordinary form of 'keps' usually consists of four legs pivoted at right angles to each other, and attached to a lever for opening or shutting them. They are generally allowed to swing out while the cage is running in the shaft, and are automatically opened by the cage itself, when it arrives at the surface, but they have to be opened by hand when the cage is about to make its descent. The commonest form of keps, made wholly of iron, is shown in fig. 261.

Stauss Keps.—In these keps (figs. 262 to 265), the invention of a German engineer, the cage is held firmly and securely in position when it arrives at the landing stage, and is released again for descent into the shaft, without the necessity of lifting a foot or two, as with the ordinary form, to allow of their being first drawn back, which often causes a sudden jerk or strain to be given to the slack rope.

When the cage is to be held fixed, it rests upon the surface *y* of the catches or tappets *c*; these catches resting upon the part *x* and against the pin *b*, and being held fast both in a horizontal and in a vertical direction. The latter function is discharged by the bell-crank *e*, which presses against the shaft *d*; whilst horizontal movement is

* *Lectures on Mining*, by Prof. Wm. Galloway, p. 7.

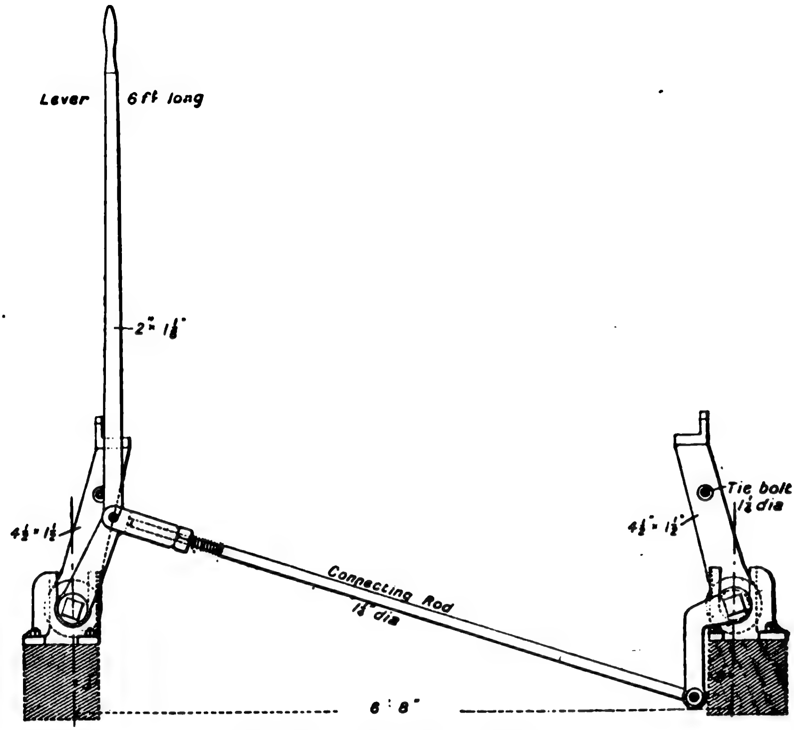
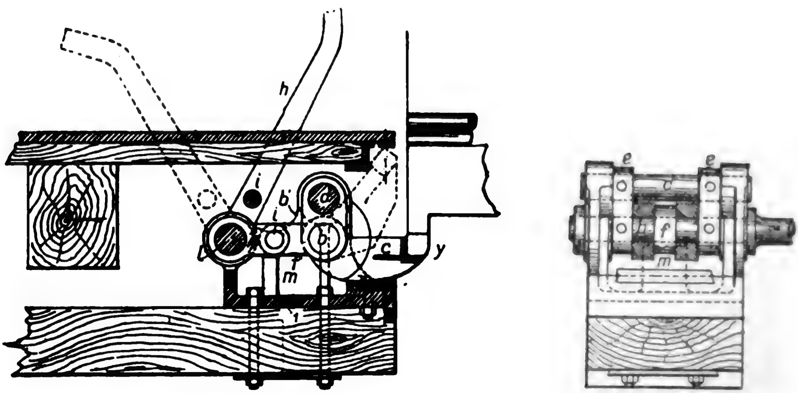


FIG. 261.—Arrangement of keps.

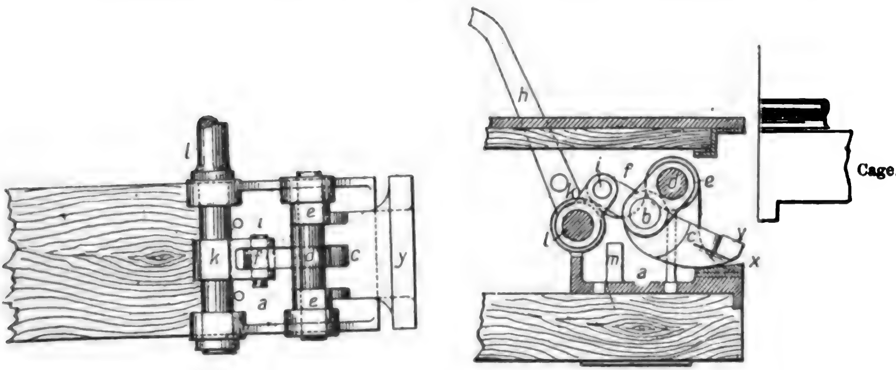


FIGS. 262, 263.—Strauss keps, shut, with cage resting.

prevented by the crank *f*, which presses against the shaft *l*, through the bolt *i*, of the lever *k*. In this way the steadiness of the cage is secured, and displacement is prevented. The hand lever *h* presses the lever *k*, when in this position, down against the block *m*, which is fixed to a casting *a*, so that side play is also prevented.

When *k* is brought into the position shown in fig. 265, and indicated by dotted lines in fig. 264, *i* is brought to the position *i′*, and *b* to *b′*, by which cage means the catches are withdrawn from under the cage and are lowered at the same time, so that the latter can descend the shaft. When the cage again arrives at the pit-mouth, the hand lever *h* is pushed back into its first position, the catch *e* projects, and the cage is secured.

The advantages claimed for these keps are: simplicity of construction and working, and a saving in ropes and engine-power, the



FIGS. 264, 265.—Strauss keps, open.

short jerks which injure the former in lifting the cage before it descends being done away with.

Hydraulic keps are used on the Continent, but they are apt to get out of order, and are not reliable, owing to their complicated nature and to the water freezing in the pipes in winter.

Fig. 266 shows the construction of these keps. Instead of four rigid arms (as in ordinary keps) there are four short cylinders, *b b*, each provided with a stuffing-box and plunger *c*. Hinged to the top of each plunger are movable pieces *d d*, which take the place of the rigid arms in ordinary keps. The four movable pieces are connected together by means of rods *e e* and levers *f f*, and move inwards and outwards like the arms of ordinary keps. The cylinders are connected to each other by a pipe *g*, common to all four, which also communicates with the cylinder of an accumulator. There is a stop-cock on this pipe which can cut off communication with the accumulator. Suppose the four plungers to be in their highest

position, and the stop-cock shut as in the figure. The loaded cage ascends the shaft, and reaching the surface pushes the four movable pieces *d d* aside, passing up between them. The latter immediately fall back, and the edge is lowered on to and arrested by them, and the liquid in the cylinders, having no outlet, prevents the plungers from descending. When the full tubs have been replaced by empty ones, the stop-cock is opened, allowing the liquid to pass into the accumulator until the movable pieces *d d* are clear of the cage.

Automatic Apparatus.—In the case of an overwind much damage is often done, which cannot be prevented even by the use of detaching safety hooks, the objects of which are to prevent the cage from falling back in the shaft. Appliances are also used to prevent loss of life if the cage happens to get overwound. In Germany every winding

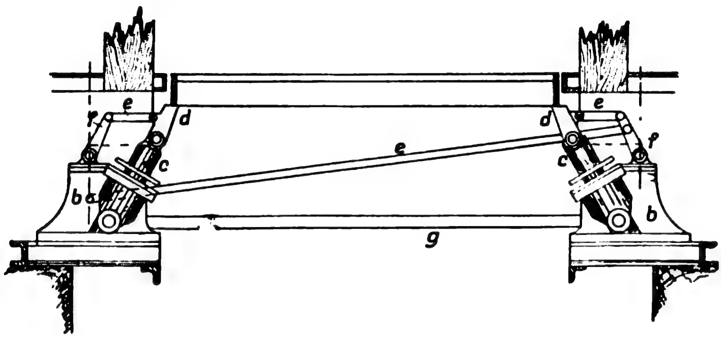


FIG. 266.—Hydraulic keps.

engine requires to be fitted with a steam brake which enables it to be brought to a standstill at once when required, even when going at full speed.

To make the brake self-acting, a hydraulic arrangement is sometimes used which the cage, when lifted too high, actuates itself. About half-way between the winding drums and the shaft, a pump, with reservoir and accumulator, supplies pressure to a length of piping leading to the shaft; and further, to a return length of piping leading back to the steam brakes on the drum. A valve at the shaft permits the pressure from the accumulator to be carried further only when it is raised by a lever, which, in case of accident, the cage itself will actuate.

The pressure thus communicated to the return length of piping acts on a vertical cylinder working on the accumulator principle, and by a rod and system of levers, may either act on the piston rod of the steam brake direct, or, by actuating the slide valves, admit the steam to the brake cylinder in the usual manner.

The Visor.—This apparatus also has been designed for the purpose of preventing overwinding, and has been in use at the pits of the Wigan Coal and Iron Co., Ltd., since 1888. The governors are driven by suitable gearing from the crank-shaft of the winding engines, as is also the worm-wheel shaft. The latter makes, approximately, one revolution per wind, and carries beaked cams, which can be adjusted to the required positions. As the speed of the engines increases, the governors rise, and move, through the medium of levers, two vertical arms with attached tappets bringing the latter into the line of revolution of the cams on the worm-wheel shaft. Towards the conclusion of the winding, if the speed of the engines is reduced suitably, the governors fall and bring the levers, with hooks attached, out of the path of the cams on the worm-wheel shaft, and into position, so that the cams pass without making contact. If through any cause the speed of the engines is not, however, suitably reduced as the cages approach the top and bottom of the shaft, the governors fail to collapse, and one of the beaked cams on the worm-wheel shaft makes contact with one of the vertical levers, the sliding frame is drawn up, the pawl raised out of the notch in the bar in the bottom of the frame, and the weight at the other end immediately falling, upsets the propped and weighted levers, and applying the steam and foot brakes shuts off the steam, and thus arrests the engines. The apparatus begins to act about two or three revolutions from the top, giving time to pull up gradually, but if the speed of the engines is maintained too long, the visor comes into operation.

Size of Winding Engine.—The calculation of the proper size and strength of the various parts of a winding engine belongs more to the province of the mechanical than to that of the mining engineer. Nevertheless at most of the examinations for colliery managers' certificates, questions on the sizes of winding engines are set, often with very insufficient data to work on, and sometimes with such as no mechanical engineer would accept.

The load which an engine has to overcome is of two kinds—viz., a 'dead load' at the beginning of the wind, and a 'live load' when the cage is in motion. It requires more force to move a dead load than to keep a live load in motion.

The simplest method of calculation is to take the work done in the shaft during one revolution of the drum, at the worst part of the wind. This will be at the moment of the cage coming to the surface, at the moment when the cage with the empty tubs has landed in the pit-bottom, and before the full cage has been brought to rest by the keps, so that the engine has the full weight to support without deriving any advantage from the descending cage.

To leave sufficient margin the load may be taken as equivalent to the combined weights of the coal raised, the rope, and the empty tubs.

The diameter of the cylinder may then be calculated from the equation,

$$D^2 \times \cdot 7854 \times P \times L \times 2 \times E = W \times \text{circumference of drum},$$

$$\text{or } D = \sqrt{\frac{W \times \text{cir. of drum}}{\cdot 7854 \times P \times L \times 2 \times E}}.$$

When D = diameter of cylinder in inches.
P = effective steam pressure in lbs. per sq. in.
L = length of stroke in feet.
E = modulus or efficiency of engine (taken at $\frac{2}{3}$ for coupled engines).
C = circumference of drum in feet (diameter × 3·1416).
W = weight of load in lbs. (coal + rope + tubs).

Example.

Calculate the size of winding engine required to draw 500 tons per shift of eight hours from a depth of 250 yds. What length of stroke, steam pressure, and size of drum would be required?

$$\text{Tons per minute} = \frac{500}{8 \times 60} = 1 \cdot 04.$$

Suppose the speed of cage is, on an average, 20 ft. per second.

$$\text{Then duration of wind} = \frac{\text{depth in feet}}{\text{speed of cage}} = \frac{250 \times 3}{20} = 37\tfrac{1}{2} \text{ seconds.}$$

Allow for time to change tubs at top and bottom (say 15 seconds).

$$\text{Total duration of wind} = 37 \cdot 5 + 15 = 52 \cdot 5 \text{ seconds.}$$

Calling this 53 seconds to avoid fractions, the load of coal raised per wind

$$= \frac{1 \cdot 04 \times 53}{60}$$
$$= \cdot 91 \text{ ton, or } 18 \cdot 2 \text{ cwts.}$$

Suppose that two tubs are raised each wind, each tub holding 10 cwts. of coal, and weighing 4 cwts. when empty; $4 \times 2 = 8$ cwts.
Take the weight of cage at three-fourths the weight of coal raised = 13·8 cwts.,

$$\text{Then coal} + \text{tubs} + \text{cage} = 18 \cdot 2 + 8 + 13 \cdot 8 = 40 \text{ cwts., or 2 tons.}$$

Then to find the circumference of rope as a preliminary to finding its weight—

$$C = \sqrt{\frac{W \times M}{3}} = \sqrt{\frac{2 \times 10}{3}} = 2 \cdot 58 \text{ in. (say } 2 \cdot 6 \text{ in.)}$$

W, in this case, being the total weight just found (2 tons).
Weight of rope in lbs. per fathom = $C^2 \times \cdot 9 = 5 \cdot 99$ (or 6 lbs. per fathom).

$$\therefore \text{Total weight} = 6 \times \frac{250}{2} = 750 \text{ lbs.}$$

Allow the effective steam pressure to be 50 lbs. per sq. in., and the diameter of drum to equal 150 × diameter of rope,

$$= \frac{2 \cdot 6 \times 150}{12 \times 3 \cdot 1416} = 10 \cdot 3, \text{ say 11 ft. diameter.}$$

Then, applying the formula given above,—

$$D^2 \times \cdot7854 \times P \times L \times 2 \times E = W \times \text{circumference of drum}$$

$$D^2 \times \cdot7854 \times 50 \times L \times 2 \times \tfrac{4}{5} = \left\{(18\cdot2 \times 112) + (8 \times 112) + 750\right\} 11 \times 3\cdot1416$$

$$D^2 \times \cdot7854 \times 50 \times L \times 2 \times \cdot8 = \left\{2038\cdot4 + 896 + 750\right\} 11 \times 3\cdot1416$$

$$D^2 \times L \times 50 \times \cdot2 = 1842\cdot2 \times 11$$

$$D^2 \times L = \frac{1842\cdot2 \times 11}{50 \times \cdot2}$$

$$= 2026\cdot42.$$

The length of stroke should be 2 to $2\frac{1}{2}$ times the diameter of cylinder. Assume a stroke of $4\frac{1}{2}$ ft.

$$\text{Then } D^2 = \frac{2026\cdot42}{4\cdot5} = 450\cdot31\,;$$

$$\text{and } D = \sqrt{450\cdot31} = 21\cdot2 \text{ in. (or 22 in.)}.$$

From the above calculation, the size of engine required would be a pair of coupled horizontal engines with

Cylinders 22 in. diameter;
Length of stroke $4\frac{1}{2}$ ft., working with an
Effective steam pressure of 50 lbs. per sq. in. ; and
Diameter of drum 11 ft. ;

the cage being taken at 13·8 cwts., and the two tubs at 4 cwts. each, with a circular steel rope, 2·6 circumference, weighing 6 lbs. per fathom.

This method of calculation is not, of course, given as theoretically correct, but it will give an approximately correct answer to the question.

Professor Merivale gives the following formula for calculating the size of cylinder when the engine is counterbalanced :—

$$A = \frac{C\left(W + \frac{W}{2}\right)}{LP}$$

Where L = twice the length of stroke in feet.
C = circumference of drum in feet.
W = weight of coal per wind in lbs.
A = area in sq. in. of two cylinders ; or half the area of cylinder if there be but one.
P = maximum pressure of steam.

Taking this formula for the above case—

$$A = \frac{11 \times 3\cdot1416 \times \left(2038\cdot4 + \frac{2038\cdot4}{2}\right)}{4\cdot5 \times 50 \times 2} = \frac{34\cdot5 \times 3057\cdot6}{225\cdot0}$$

$$= 234\cdot41 \text{ sq. in.}$$

$$\therefore D = \sqrt{\frac{234\cdot41}{\cdot7854}} = 17\cdot3 \text{ in., } \textit{diameter of cylinder.}$$

The size of winding engine required can also be worked out in the following way :—

Example.—What diameter of cylinder would be required to raise 1000 tons of coal from a depth of 200 fms. in eight hours; the tubs 5 cwts. each, and carry 14 cwts. of coal, 4 tubs being raised on each cage, the latter weighing 40 cwts.? The effective steam pressure is to be 60 lbs. per sq. in., the stroke of engine 5 ft., and the diameter of the winding drum 16 ft.

Tons per hour $= \frac{1000}{8} = 125$, winds per hour $= \frac{125 \times 20}{4 \times 14} = 44\cdot6$, or 45 for convenience. Time per wind $= \frac{60 \times 60}{45} = 80$ seconds; and assume the time taken to

change the tubs is 20 seconds, then the actual time occupied in winding will be $80 - 20 = 60$ seconds, and as the shaft is 1200 ft. deep, the average speed of cage will be $\frac{1200}{60} = 20$ ft. per second. The maximum velocity will be about double this, or 40 ft. per second. The time in which this velocity is obtained may be taken as $\frac{1}{7}$th of the total time of winding, or 8·55 seconds.

The circumference of rope required.*

$$= \sqrt{\frac{L}{\frac{4}{M} - \frac{D}{1 \cdot 1 \times 2240}}} = \sqrt{\frac{5 \cdot 8}{\frac{4}{10} - \frac{200}{1 \cdot 1 \times 2240}}} = \sqrt{\frac{5 \cdot 8}{\cdot 4 - \cdot 081}} = 4 \cdot 25 \text{ in.}$$

Weight of rope per fathom $= C^2 \times \cdot 9 = (4 \cdot 25)^2 \times \cdot 9 = 16 \cdot 3$ lbs. If the head gear is 60 ft. high, then the total weight of rope is $210 \times 2 \times 16 \cdot 3 = 6846$ lbs.

Let W = weight to be set in motion; two cages, coal, empty tubs on cage, two winding ropes from pit-head pulleys to pit-bottom.
- V = greatest velocity obtained, uniformly accelerated from rest = 40 ft. per second.
- g = gravity = 32·2.
- t = time in seconds during which V was obtained = 8·55 seconds.
- L = unbalanced load on engine = coal.
- P = effective steam pressure in cylinders = 60 lbs. per sq. in.
- N = number of cylinders = 2.
- s = space passed through by crank pin in time t.
- $c = \frac{2}{3}$ constant to reduce angular space passed through by the crank to distance passed through by the piston during time t.
- A = area of cylinder in square inches.
- D = diameter of cylinder required.
- f = allowance for friction of engines, etc., taken at 20 to 30 per cent.

(*a*) The greatest work the winding engine has to do is to get the mass W into a certain velocity, uniformly accelerated from rest. This work is difficult to calculate properly, for to do so the energy required to set the winding drum and pulleys in motion would have to be accurately ascertained. To allow for this energy to move these parts, the mass W has been taken to include the weight of the two winding ropes and the two cages.

$$\text{Resistance due to gravity and inertia} = \frac{WV}{gt}.$$

$$\text{Work done in overcoming resistance} = \frac{WV}{gt} \times \frac{Vt}{2} = \frac{WV^2}{2g}.$$

(*b*) To raise the unbalanced load L, distance passed over in time t.

$$\text{Work done in ft. lbs.} = L \times \frac{Vt}{2} \quad \therefore A = \frac{\frac{WV^2}{2g} + L\frac{Vt}{2}}{P \times S \times N \times C}.$$

If the load is balanced, $L = 4 \times 14 \times 112 = 6272$ lbs., and $W = 24{,}318$ lbs. In time t the drum will make about 2·5 revolutions, $\therefore S = 2 \cdot 5 \times 5 \times 3 \cdot 1416 = 39 \cdot 25$.

$$A = \frac{\left(\frac{24{,}318 \times 40^2}{2 \times 32}\right) + \left(6272 \times \frac{40 \times 8 \cdot 55}{2}\right)}{60 \times 39 \cdot 25 \times 2 \times \frac{2}{3}} = 535 \cdot 17.$$

Allowing 20 per cent. for friction, $A = 535 \cdot 17 + 107 \cdot 03 = 642 \cdot 20$, and $D = \sqrt{\frac{642 \cdot 20}{\cdot 7854}} = 28 \cdot 4$ in. The size of the winding engines would therefore be 28·4 in. diameter with a 5-ft. stroke.

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* See formula for calculating size of winding ropes, pages 217, 218.

CHAPTER XI.

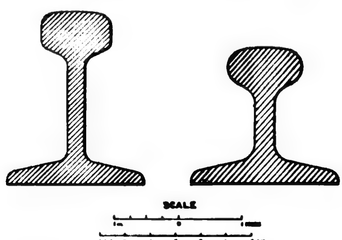
HAULAGE.

Classification of Methods.—The different modes of haulage employed underground may be classified as follows:—(1) Manual labour; (2) horse haulage; (3) self-acting inclines; (4) mechanical haulage by stationary engines placed either at the surface or underground; (5) electrical or compressed air locomotives.

Haulage Roads.—There is hardly anything of greater importance to an effective system of underground haulage than the laying down of the road. A good road when properly constructed should last as long as the colliery without entailing a great deal of repair or expense. On the other hand, a badly arranged and badly laid road will cause continual trouble and anxiety and entail great expense for repairs, so that it will cost far more in the long run than a good road, to lay down which may, at first, be a little more expensive.

Rails.—The different rails in use are of three patterns, the flat-bottomed made of cast iron or steel, the 'bridge' rail made either of wrought iron or steel, and circular-head rails usually made of steel.

Flat-bottomed rails made of cast iron with side flange were, at one time, almost universally employed, but the pattern of tub wheel required with this description of rail occasioned so much friction, that it is being almost entirely abandoned in favour of the newer and more efficient circular-head or T-rail, except in old collieries where there are large stocks of old rails which it is desirable to use up. The newer kind of rail gives the minimum of friction, is easily laid, and, when made of steel, lasts a long time. It generally weighs from 14 to 20 lbs. per yard according to the size of tub used and the weight of load. For main haulage roads, the heavier sections should be selected.



FIGS. 267, 268.—Section of rails.

Friction of Rails.—The force required to overcome friction is usually estimated at—

8 to 10 lbs. per ton on a surface railway ($\frac{1}{224}$).
32 ,, edge of T-rails underground ($\frac{1}{70}$).
70 ,, flat-bottomed or tram-plate rails underground ($\frac{1}{32}$).

Method of Laying Roads.—Where a large amount of coal has to be hauled over a road, the rails ought to be well laid, on some such system as on a surface railway, with joints 'fish-plated' and rails well keyed and with plenty of 'chairs.'

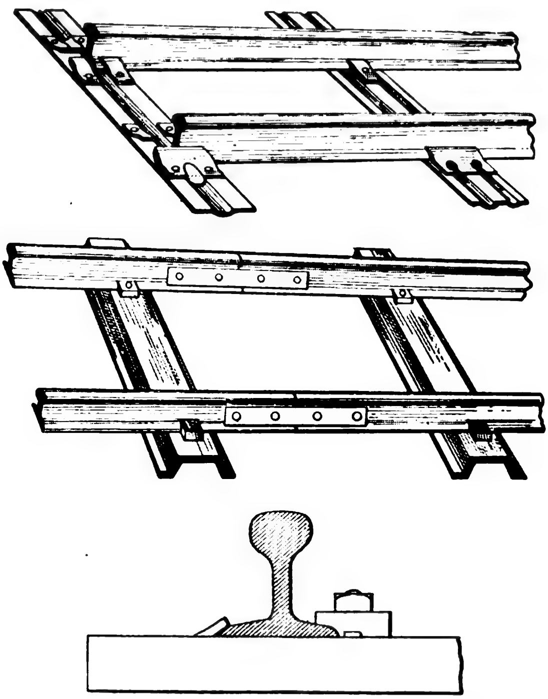
On the best haulage roads the rails are sometimes laid on longitudinal sleepers, which are held together by cross sleepers, with the joints of the rails fish-plated. The longitudinal sleepers are often ordinary white pine planking 9 in. × 3 in., and the cross sleepers 5 in. × 1½ in. Wrought-iron and steel sleepers are also used in place of the ordinary wooden sleepers. Laying roads with iron or steel sleepers naturally costs a good deal more than when wood is employed, but the greater durability and stability more than compensate for the increase in first cost. Figs. 269, 270, 271, show the method of laying such roads, and will require no further explanation.

Gauge.—The gauge will depend upon several considerations, such as whether the wheels project beyond or are under the body of the tub, the inclination of the seam, etc. It may vary from between 18 to 36 in., 24 in. being a common gauge. Narrow gauges are most suitable for flat seams, but where the inclination is great the gauge should be increased in proportion. The writer has seen a gauge of 36 in. employed where the inclination was between 30° and 45°.

Tubs.—These are variously called 'trams,' 'corves,' 'hutches,' or 'tubs,' in the different mining districts. The body is usually rectangular in shape, but sometimes they are semicircular at the bottom, which increases their capacity for a given height. The design and size of tubs used will be governed by the varying conditions under which they are required to work, such as the thickness of seam, height and inclination of roads, and whether manual or other kinds of haulage are most largely used. For thin seams where manual labour is used to any considerable extent for haulage the tubs should be made of small capacity and weight. A tub to hold 10 to 12 cwts. should not weigh more than 4 cwts. if constructed of wood, or 5 cwts. if of iron. The capacity varies greatly, and may be anything between 5 and 40 cwts., but an average size is from 10 to 15 cwts. In the South Wales coal-field tubs holding 30 to 40 cwts. are often used, while in some of the thin seam collieries of Somersetshire, and in Scotland, tubs holding 7 to 9 cwts. of coal are common. The advantage in having large tubs is that fewer windings at the pit shaft need be made per diem for a given output, and that their capacity is large in proportion to their weight. On

the other hand, they are clumsy to handle, and, if derailed, are difficult to place on the rails again. On the whole, a tub of medium capacity, *i.e.* 12 to 15 cwts., is to be preferred, as it can be moved about easily and lifted on to the rails by one man when necessary.

Materials of Construction.—Tubs may, as has been stated, be constructed either of wood, wrought iron, or steel. Opinions differ among mining men as to which of these materials is best. For light loads wood is very commonly used, and it has the advantage of

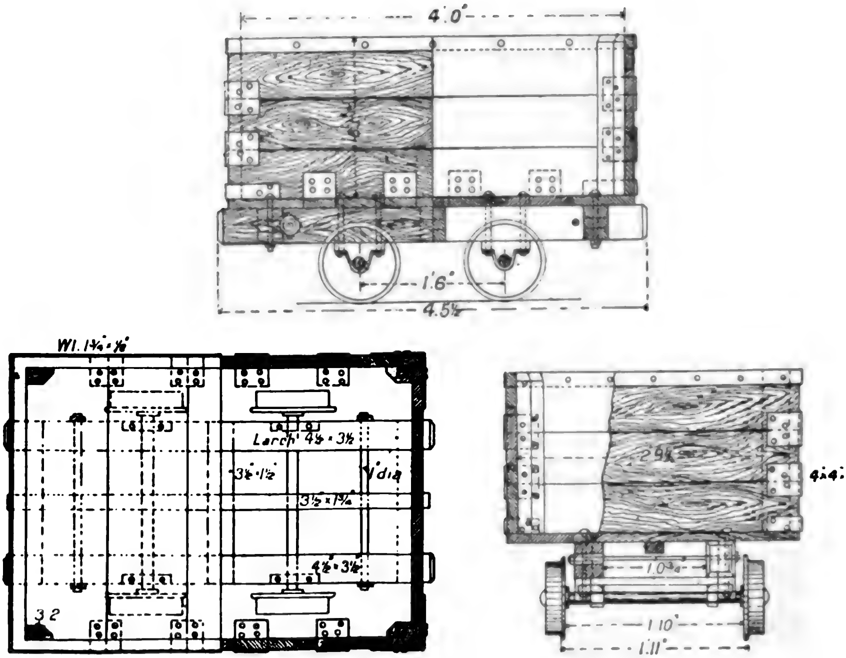


FIGS. 269, 270, 271.—Method of laying rails.

being cheap and light, while the cost of maintenance and repair is less than for steel, and less skilled, *i.e.* cheaper labour is required for constructing and repairing wooden than for iron or steel tubs. Again, it not infrequently happens that even in the best regulated collieries a train of tubs breaks away on inclined haulage roads, and when such an accident takes place, the tubs will be more or less smashed and broken. If made of wood they can soon be repaired, even although they may be badly damaged, whereas if constructed of wrought iron or steel they would be so twisted and

bent that the work of a skilled blacksmith would be required for their repair. In thick seams lying at a comparatively small inclination, iron or steel tubs are preferable, especially if the load is large (20 to 40 cwts.).

Figs. 272, 273, 274 show the construction of a tub designed by the author and built of wood. The framework is made of two 'trams' of larch wood $4\frac{1}{2}$ in. × $3\frac{1}{2}$ in., held together by oak 'starts' $3\frac{1}{2}$ in. × $1\frac{1}{2}$ in., and further strengthened by two iron rods 1 in. diameter. The body is of larch cleading $1\frac{1}{8}$ in. thick, bound at



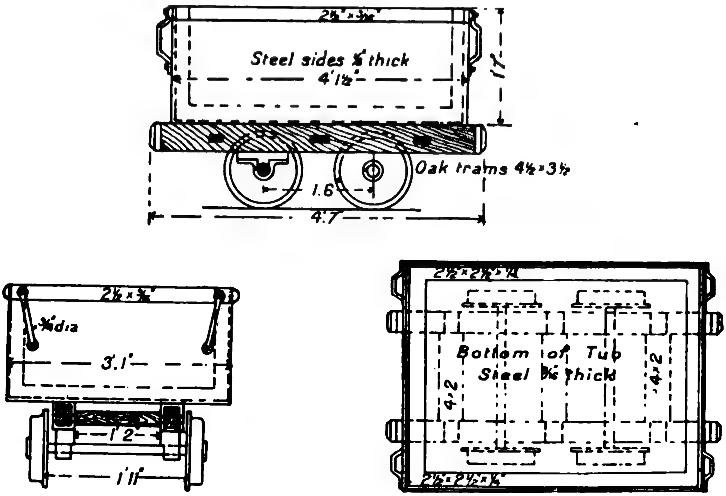
Figs. 272, 273, 274.—Details of construction of wooden tub.

the sides and ends with pieces of sheet-iron angle $\frac{1}{8}$ in. thick. It is also bound round the top with a band of wrought iron. The capacity is $9\frac{1}{2}$ cwts. when filled to the level of the sides, and about 12 cwts. when heaped up above them. The cost of such a tub would be about £2, 15s. or £3 complete.

Figs. 275, 276, 277 show a tub constructed of steel sides and ends resting on a wooden frame. The latter is of larch held together by oak 'starts' 4 in. × 2 in. The sides are of steel $\frac{1}{8}$ in. thick, and angle iron $2\frac{1}{2}$ in. × $2\frac{1}{2}$ in. × $\frac{1}{4}$ in. running round the bottom, and also at each corner, in order to bind the sides and ends together.

The capacity of such a tub is 10 cwts., or 12 to 14 cwts. when heaped. The estimated cost is £4, 10s.

Arrangement of Pit-bottom.—The first consideration in any system of haulage is to have the pit-bottom laid out so as to best accommodate the tubs as they come and go. Many pit-bottoms are too confined, which may be a saving in first cost, but is never satisfactory in working. It should be so arranged that all the empty tubs can be taken off the cages at one side and the full tubs



FIGS. 275, 276, 277.—Construction of steel tub.

placed on at the other. Where it can be carried out, this is the most satisfactory arrangement.

Figs. 278, 279 show the arrangement of the pit-bottom at Earnock Colliery. Here the coal is raised from two levels, and these are so arranged that both decks of the cage can be charged at the same time, without the cage being moved. Fig. 278 shows a section of the shaft which more clearly explains the working. On the top deck, Ell coal is caged, while the lower deck is reserved for coal from the Main and other seams. To carry this out the lower level had to be formed in the strata above the Main seam, and a stone mine driven at an inclination in order to catch the coal. Fig. 279 shows the plan, which will be readily understood. On both sides of the pit-bottom there are double roads for full tubs and a road for empties. On the side A, which is worked by main and tail rope, the road for the empties dips away from the shaft for a certain distance at a gradient of 1 in 50 and then rises to the level of the other road at an inclination of 1 in 30, so that no labour is entailed

in pushing the empty tubs forward to the desired position. The two full roads on each side are constructed to hold about 80 tubs. From the high to the low stage a by-road is formed, so that if there is not sufficient coal coming in to keep the low deck going, a train of tubs can be run down from the high level A to the low level B. The whole arrangement works very well, over 1200 tons being often raised in eight hours, with four tubs on each cage. Of course this arrangement would not suit in every case, but it may be taken as an illustration of a well-planned pit-bottom.

Haulage.—*Manual labour.*—The method of haulage by men and boys can only be employed with advantage where the drawing roads are of moderate length and comparatively level, and where the tubs are small. If the length of road is great, or the inclination high, it is the most expensive system that can be adopted, and in any case

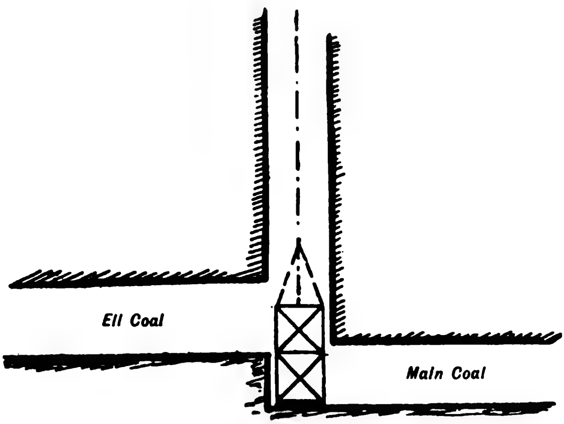


FIG. 278.—Elevation of pit-bottom.

it is always better, where the height of seam will allow, to bring the tubs direct from the face by means of ponies or by some other method of traction. Manual haulage should therefore be confined to short distances on level roads. In some small collieries where the seams are thin and the roads flat, it is often the only method employed for hauling the coal from the faces to the pit-bottom.

Horse Traction.—Haulage by horses is employed more or less in nearly all collieries, there being sometimes 100 to 150 horses underground. If the inclination of the roads is not too great, it is often an economical and convenient form of haulage, especially if there is a slight inclination in favour of the load towards the pit-bottom. A horse is capable of exerting a tractive force of 120 lbs., when travelling at the rate of two to three miles per hour, and can keep this up for a period of ten hours, which enables it to draw 24,000 lbs. or

10 tons for a distance of twenty miles, or 200 tons for a distance of one mile per diem. Two-thirds of this estimate is a fair average of the work which a horse usually performs. The mileage depends greatly on local circumstances, but on moderately level roads horses ought to travel fifteen or sixteen miles per day, and in roads dipping 1 in 20, eight or ten miles per day, if the roads are, in each case, in good order, with long runs and few stoppages.

FIG. 279.—Plan of pit-bottom.

Horse Roads.—The most economical form of horse road is where the work done, in drawing in an *empty load*, is equal to the work done in taking out a *full load*; which for edge rails should be an inclination of about 1 in 140, or $\frac{1}{4}$ in. per yard, in favour of the full load.

When F = friction of full load,
f = friction of empty load, and
G and g = gravity due to full and empty loads respectively,

$$F - G = f + g.$$

If I = fraction representing gradient,
H = height of incline above horizontal,
L = length of incline,
W = weight of full load,
w = ,, empty load,

$$I = \frac{H}{L} \text{ or } G = \frac{WH}{L};$$

$$\text{and } F - \frac{WH}{L} = f + \frac{WH}{L} \text{ and } \frac{H}{L} = \frac{F - f}{W + w},$$

$$\text{but as I also} = \frac{H}{L} \therefore I = \frac{F - f}{W + w}.$$

Example.—What should be the gradient for a horse haulage road, if the load consists of 12 tubs, each holding 15 cwts., and weighing 5 cwts. when empty, allowing $\frac{1}{70}$ for friction?

$$I = \frac{\left(\frac{12 \times 15}{70}\right) - \left(\frac{12 \times 5}{70}\right)}{(12 \times 15) + (12 \times 5)}$$

$$= \frac{\frac{180}{70} - \frac{60}{70}}{180 + 60} = \frac{\frac{180 - 60}{70}}{240} = \frac{1 \cdot 71}{240} = \frac{1}{140},$$

or 1 in 140 (the gradient required).

Feeding of Horses.—This forms one of the heaviest items in the keeping of horses underground, as a fair-sized horse will cost 10s. to

12s. per week for food and bedding alone. The aim is to keep horses in the best possible condition at the minimum of cost, and this can be easily accomplished if the right feeding-stuffs are selected. Formerly the feeding consisted chiefly of boiled food, composed largely of beans, barley and bran, along with dry hay. Now it has been pointed out by qualified authorities, that bran has little or no feeding value unless as a laxative, and that barley and beans should only be used sparingly. Nearly all collieries now adopt dry feeding, or 'chop,' as it is termed, for the horses, this being found to keep them in harder and better condition than soft or boiled food. The latter, however, has its place, as animals, like men, appreciate a change, and this may be given at least once a week with advantage. Some green food during summer will also be relished, and is beneficial. The following 'chop' is sometimes given :—

Cut hay,	8 cwts.
Oats,	5 ,,
Beans,	2½ ,,
Indian corn,	2½ ,,
Peas,	2 ,,
Total,	20 ,,

Mr J. B. Hamilton, in a paper read before the Mining Institute of Scotland, gave the following mixtures for a daily feed :—

	No. 1.	No. 2.	No. 3.
Beans or peas,	2 lbs.	2 lbs.	...
Maize (bruised),	6 ,,	4 ,,	4 lbs.
Oats,	4 ,,	3 ,,	3 ,,
Bran,	...	2 ,,	2 ,,
Hay (cut),	12 ,,	12 ,,	12 ,,
Straw (cut),	2 ,,	2 ,,	2 ,,
	26 ,,	25 ,,	23 ,,

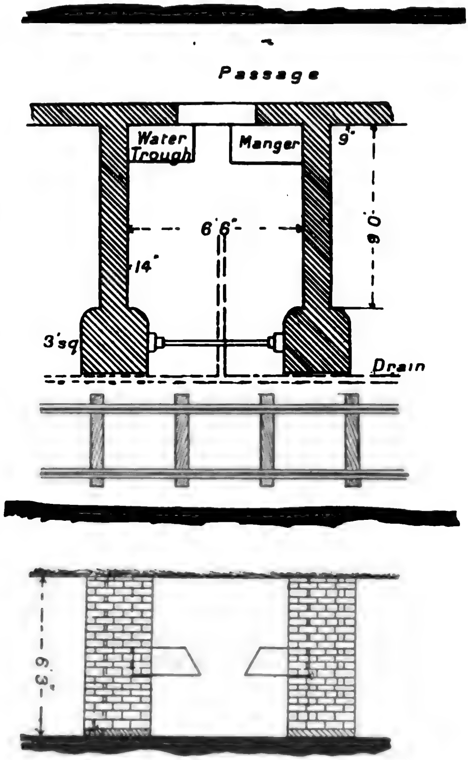
No. 1 is a daily feed for a pony about fifteen hands high hardened to its work; No. 2 for same size of pony new to work; and No. 3 is for a pony doing no work.

Mr Hunting * says that the quantity of food must be regulated by the amount of work required; but about 100 lbs. of corn, crushed and mixed with about 56 lbs. of chopped hay, will form an average week's provender for each horse. He considers that beans (or peas) and hay with either (1) oats and bran, (2) barley and bran, (3) oats and maize, (4) maize, will form equally good mixtures, and that we must be guided in our selection by current prices.

Feeding is not, however, the only thing to which attention should be devoted. Keeping horses in good condition requires a plentiful supply of good clean drinking water, and they must also be properly bedded, and above all, kept well cleaned and groomed. It is too often left to the pony driver to attend to these matters, but no

* *Trans. N. Eng. Min. and Mech. E.*, vol. xxxiii. p. 61.

greater mistake can be made, especially if there are sufficient ponies to afford work for an ostler, as many pony drivers are mere boys, who have no experience whatever as to how a horse ought to be cleaned, and the majority of them are too careless to pay much attention to the needs of the horse or pony under their charge. Wherever there are more than a dozen horses underground, it will, as a rule, pay to put them under the charge of a competent ostler or horse-keeper. Horses should also have an opportunity of eating during working hours, as their work is exhausting, and a horse cannot retain sufficient food to maintain it for long intervals. Provision should therefore be made at the lyes or sidings, where the horses stop, for them to obtain food while waiting. A visiting veterinary surgeon should also be appointed, where there are a large number of horses, to examine them periodically, and to report upon their condition. Horses underground ought to be properly shod at stated intervals. The cost of shoeing will on an average amount to about 6d. per week. The total cost of each horse per week may be estimated at 12s. to 15s., including shoeing, attendance, repairs to harness, feeding, and bedding.



FIGS. 280, 281.—Plan and section of stables.

Stables.—The arrangement of the stables is a very important matter, and it will pay to expend some thought and care in planning them. Many stables are merely formed in the workings, without any attempt at flooring or drainage, the result being that the floor gets worn into holes, making it very uncomfortable for the animals to rest on.

Figs. 280, 281 show the construction of stables as sometimes

adopted. The stalls, which are walled in on each side with a brick wall 14 in. thick, are 9 ft. × 6½ ft. × 6 ft. A gutter is formed down the centre of the lower half of the stall, and connected with a main gutter running at right angles to it, and in the direction of the roadway. A tram road is laid along between the rows of stalls, for convenience in cleaning them, and a passage is also constructed along the head of each stable for the distribution of food. The floor should be well laid with bricks, laid sideways, and carefully cemented in, the floor having a rise towards the head of about 2 in. in the yard. A large watering-trough should be placed outside the stables for the horses to drink, when going to and returning from work.

Cost of Horse Haulage.—The cost of horse haulage depends on various conditions, such as the gradient of the road, the condition of the rails, roads, and tubs, especially the wheels, and whether the roads are wet or dry. The cost of horse haulage at the faces, *i.e.* lifting tubs from the faces to mechanical haulage, may be taken as 9d. to 1s. 6d. per ton per mile; the cost of horse haulage when used for conveying coal from lyes to the pit-bottom, in rakes or sets, may vary from 2d. to 6d., or sometimes as high as 1s. per ton per mile. The cost for keep of horses, shoeing, harness, etc., has to be added. This may be taken at ½d. to ¾d. per ton per mile. Prof. William Galloway gives the following details of the cost of secondary haulage, *i.e.* collecting tubs from working places and conveying them to sidings:—

System.	Agents.	Unit of Weight.	Day's Work.	Cost per Day.		Cost per Ton per Mile.	
		lbs.	yard, tons.	*s.*	*d.*	*s.*	*d.*
Handputter,	Boy, . .	550	1610	2	0	2	2·3
Horse, .	Man and horse, .	2800	6864	6	6	1	8
Pony, . .	Lad and pony,	896	5577	4	5½	1	4·8
Hand, .	Man, . .	896	5252	3	8	1	2·74

In these examples the gradient of the roads where the different agents are employed will vary considerably.

Self-acting Inclines.—Self-acting inclines may be divided into three different classes:—(1) Balance braes or jig brows, where a balance weight is used. (2) Self-acting incline working with two tubs on the cut-chain principle. (3) Self-acting inclines working in the ordinary way with trains of tubs.

In the great majority of mines, the conditions are such that very often one or all of the above sorts of incline can be employed. Where the inclination of the seam is sufficient to allow them to run freely, they form very easy and economical means of haulage.

Balance Brae or Jig Brow.—This is the simplest form of self-acting

incline, and is usually employed at the faces for bringing out a single tub at a time, to a siding or main incline. It is worked by a rope or chain, more usually the former, passing round a wheel, 18 in. or 24 in. diameter, fixed between two iron jaws which terminate in a screwed bolt passed through a prop at the face, and held tight by a nut and washer (fig. 282). Two sets of rails are provided, one set having a narrow gauge for a loaded carriage or bogie to run on, and the other set with wide gauge for the tub to run on. The bogie is filled with scrap iron, and is generally so balanced that it can bring up the empty tub without requiring any brake on the wheel. The full tub on descending draws the loaded bogie up to the face, and the bogie on descending brings up the empty tub. To prevent the loaded carriage from running back, when the full tub is disconnected from the rope, at the foot of the incline, the rope is held fast as shown in fig. 283. To the short piece of chain attached to the capping of the rope, an auxiliary piece is fixed. To a double sleeper is fixed a bolt over which a link

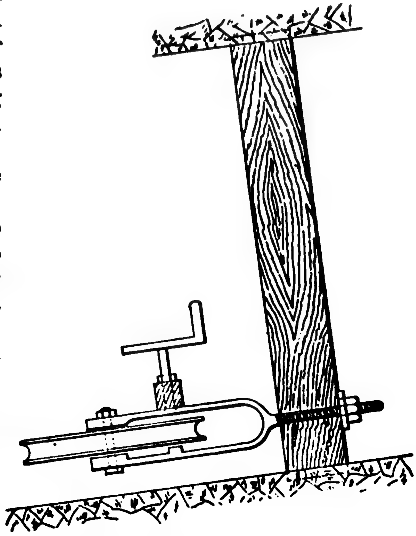


FIG. 282.—Jig brow arrangement.
(It is understood that the floor line inclines about 10° to 15°.)

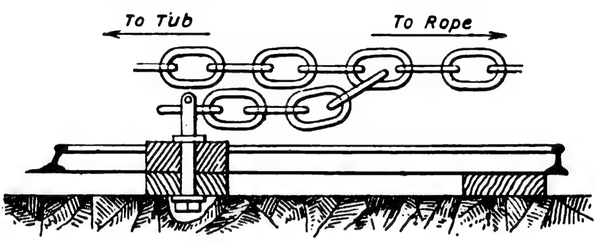


FIG. 283.

of the auxiliary chain is passed, and held in position by a pin. This holds the rope in position until the empty tub is attached.

Jigs are best suited for seams where the gradient is greater than 1 in 6, and the author has seen them working satisfactorily in roads

having an inclination of 35° to 40°, but in such cases great caution must be exercised, and the roads should be thoroughly well laid. In highly inclined roads the wheel at the face ought to be provided with a hand brake as shown in the sketch, and the prop well notched into the roof and floor. Unless this is carefully attended to, accidents are almost sure to occur.

Cut-chain Incline.—This is a somewhat different form of self-acting incline to the above, for instead of a loaded carriage being used, a

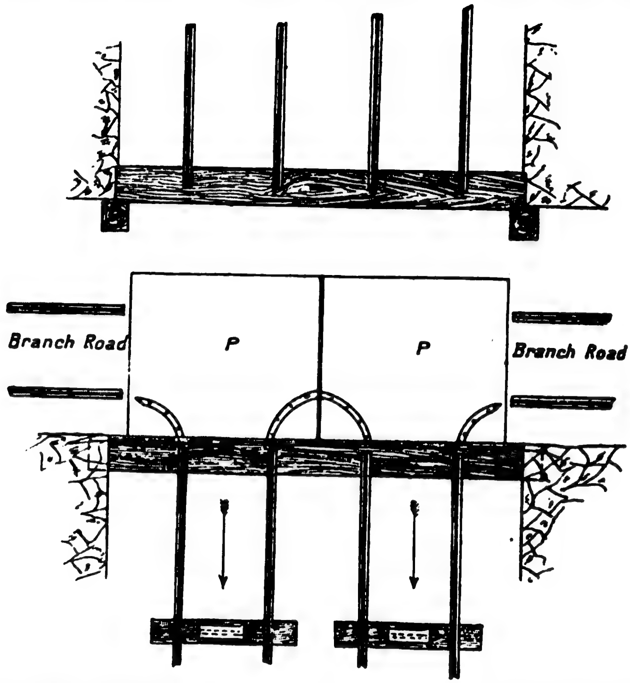
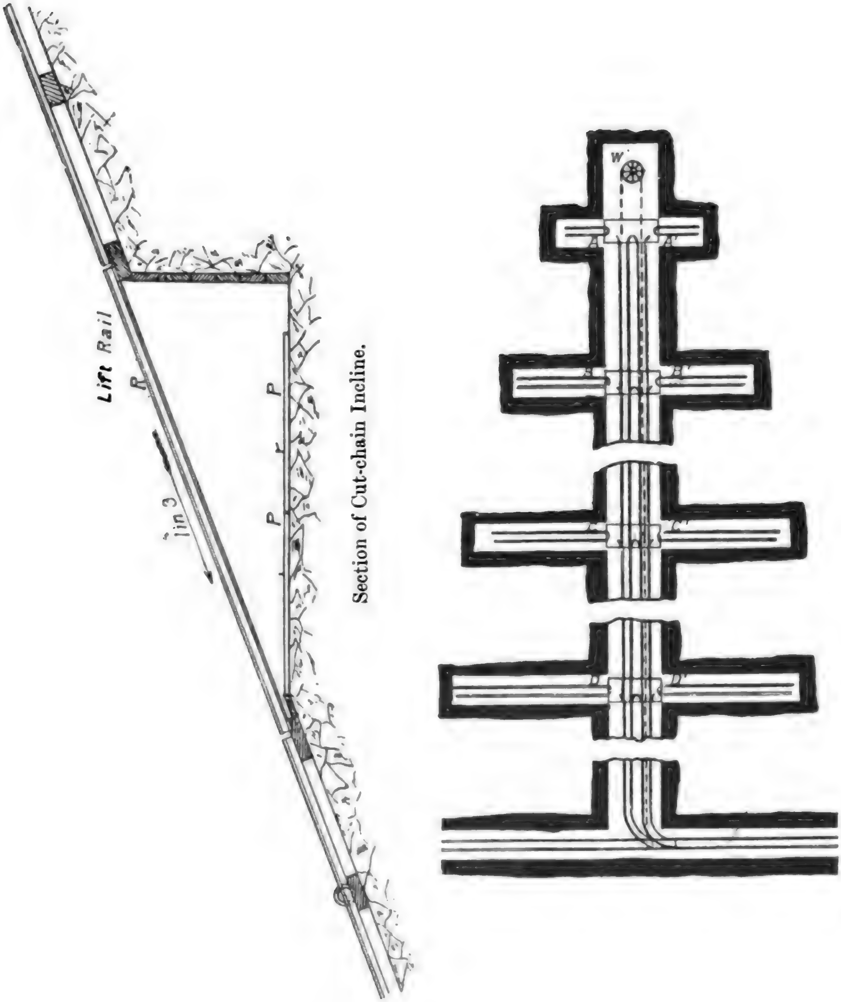


FIG. 284.—Enlarged plan of cut-chain, showing main and branch roads, with lift rails out of position.

regular double road is required, and the weight of the full tub going down brings the empty tub up to the face. This system is largely used in Fifeshire and other districts of Scotland, and can be employed either in stoop and room or longwall working, if the inclination is sufficiently great. The working of this system will be understood from figs. 284–286.

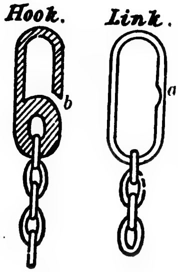
Method of Working the Cut-chain Incline.—Suppose a tub requires to be run down from A, the portion of the chain not required is detached and left lying on the road between the rails. The full tub is then attached, run down, and an empty one brought to A′; another

The rope passes round a wheel at the top, W, fig. 286, and has connections or 'cuts' at each of the 'hanging on' stages, A B C D, at none of which are any platforms used, but the road is levelled out and two plates P P, figs. 284, 285, with lift rails R R, pass over them to the rails on the opposite side. Fig. 286 shows a plan of how these branches are arranged.



Figs. 285, 286.—Cut-chain incline.

full tub should be kept ready at A′ to be run down and bring up an empty to A, so that the chain can be again connected and ready for a 'cut' at any of the other branches B B′, C C′, D D′. At the parts where the chain is cut, special links are required for disconnecting and reconnecting the chain. These links are shown in figs. 287, 288. The thin part *a* of the link in fig. 288 is made so that it fits the link in fig. 287 through the opening *b*. When the two links are thus connected, it is impossible for them to be separated, as no other part of the link except the part *a* can pass through the opening *b*.



FIGS. 287, 288.

At each branch road, highly inclined (30° to 45°) from the horizontal, the roadway is levelled out and cast-iron plates laid down, with 'lift' R rails to pass over them to connect the rails above and below the branch. When a tub is being run down from a branch road, these 'lift' rails are lifted out of position and laid aside until the full tub is taken down and the empty one brought up, when they are again replaced, which leaves the road joined up ready for tubs to be run from any of the other branches above. If the gradient does not exceed 1 in 4 or 1 in 3, the lift rails are usually dispensed with, the floor of the road at the branches simply being levelled out somewhat, and the plates laid down with a gradient of 1 in 15 to 1 in 20, and the tubs allowed to run over them without the aid of the 'lift' rails. To enable the tubs to pass easily over these plates, they are cast with a groove the gauge of the wheels, so that when the tub leaves the rails the flanges of the wheels enter these grooves, in which they run until the rails are again entered upon below the branch.

When branch roads are worked from one side of the heading or incline, the arrangement for running the tubs is somewhat different.* A wheel is fitted up at the top of the heading in the way already described, and the chain passed round it, one end reaching the foot of the incline and the other lying near the top. At each branch road or 'cut' there is a disconnecting or cut-link in the chain as in the first arrangement, so that the upper portion can be disconnected as required. At the branch roads plates, with grooves in them, are laid down in the same same way as shown for the incline, with branches worked from both sides. At a short distance above these plates, about 6 ft. or so, a small grooved wheel is fixed, preferably between the two lines of rails. Whenever a tub requires to be run down from any of the branch roads the chain is disconnected at the cut-link, and the end of the lower portion passed round this small pulley and connected to the end of the load tub. To prevent the tub from running off before the chain is attached, a block made of a piece of wood 4 or 5 inches square and of sufficient length to stretch across

* *Trans. Inst. Min. Eng.*, vol. xiv. p. 196.

both rails, is fixed across them. This block turns on a bolt at one end, and when in position is fixed by an iron pin at the other end, which passes through the block into a plank or sleeper on the roadway. When the end of the chain has been drawn round the wheel and fixed to the tub, the latter is run down to the foot of the incline, an empty tub being brought up on the other line of rails to the branch road. Like the arrangement previously described, two tubs should be run from each branch so as to bring the chain back to its original position ready for connecting. If this is not done, and only a single tub is run down, the position of the chain reaching from the branch road, from where it has been 'cut,' will have to be lifted across to the centre of the line of rails where it was originally, and this can only be done if there are no centre props on the incline. On the wheel at the top, if the gradient requires it, a brake is fitted, operated from each of the branches or benches by means of a wire attached to the brake-handle, and led down the whole length of the incline.

A large amount of coal can be run down these

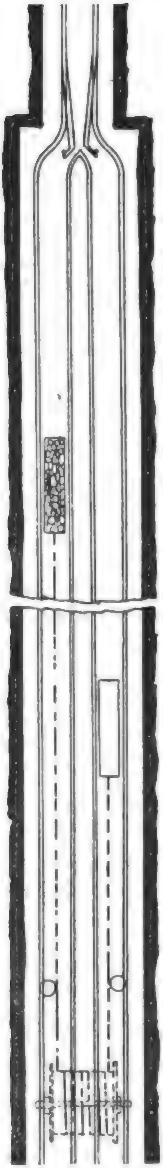


FIG. 289.—Self acting incline.

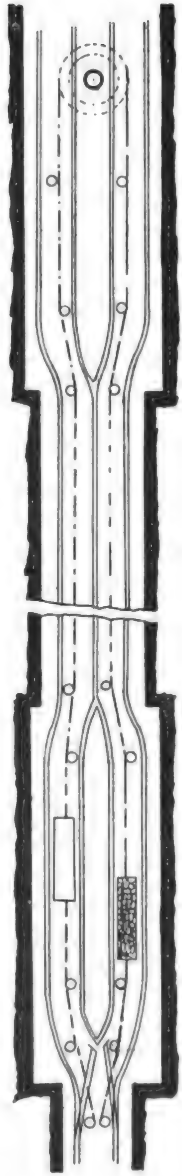
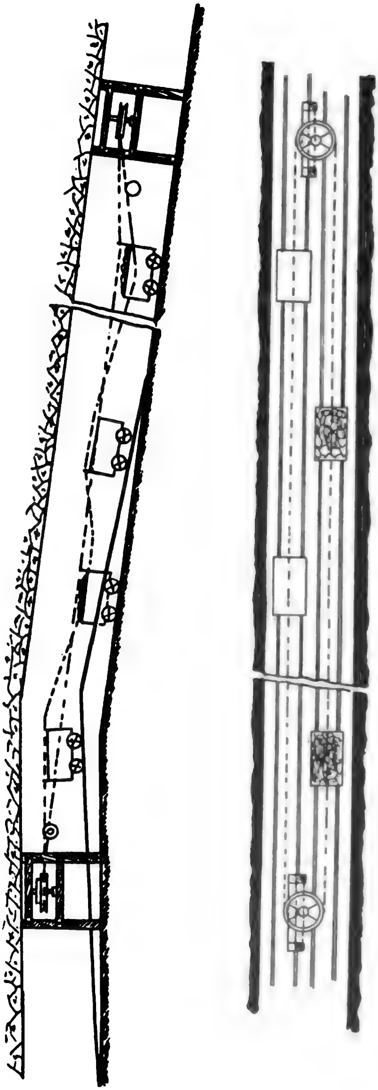


FIG. 290.—Incline with double tram lines.

inclines when they are properly constructed. They are best suited for lengths of 50 to 100 fms., beyond which they do not work so well.



Plan and Elevation of Incline.

Figs. 291, 292.—Endless chain incline.

The same system can also be used with a balance weight in long-wall working where it is often difficult to keep the road wide enough for a double tram road. The cut-chain system can be worked on any inclination from 10° up to 45°.

Self-acting Incline with Trains of Tubs.—This is the commonest form of incline, and can be used with great advantage where the gradient is suitable. The most usual arrangement is to have a rope passing two or three times round a drum or wheel, to give sufficient friction and prevent slip. The drum or wheel is fitted with a good brake, and the rope is attached to the full train of loaded tubs, which by their own weight bring a train of empties to the top of the incline. The best plan is to have a double road throughout (fig. 289), as the incline will work more satisfactorily than by using two or three sets of rails. If the roof is bad it may be difficult to make and maintain a road of sufficient width to admit of a double train line throughout. In such circumstances two or three rails may be used with a pass-by in the centre. Fig. 290 shows an incline with three rails and pass-by.

If the inclination is great the best results are got by a drum instead of a wheel for lowering the tubs, as it can be better

kept under control; on the other hand, the ropes are more difficult to keep in line with the roads, owing to the lapping of the ropes on the drum. If the inclination is not too great a suitable wheel gives very good results, and has the advantage of taking up little space, and the ropes can be kept in a straight line with each road, if the wheel is fixed horizontally. About 3° is the least inclination an incline will work at easily, but the length of incline is a determining factor, as the longer the incline the greater will be the weight of rope and the friction. On the other hand, the shorter the length of incline and the heavier the load, the better will it work.

Self-acting inclines are sometimes worked on the endless rope or chain principle, with the tubs attached singly at stated distances apart. Where the conditions are suitable this plan yields very good and economical results. Figs. 291, 292 show the arrangement of such an incline worked with an endless chain, and fig. 293 illustrates the brake wheel used on this kind of incline, and which is somewhat differently arranged to an ordinary incline wheel, as the attendant has to handle the brake, and also to attach the tubs to the chain. The tubs are attached to the rope at intervals of 20 to 30 yards; where a tub has to be attached the attendant applies the brake and brings the rope to a standstill; the tub is attached, and the rope is allowed to move until the next interval. The advantages claimed for this system are: small cost for upkeep of rolling stock, the slow speed causing fewer breakages; regularity of delivery; and economical working. The length makes little difference, and there is a little less expenditure in making benches at the top and bottom of the incline. Although roads with regular gradients are best adapted for the application of this system, it can also be successfully employed upon roads the inclination of which varies.

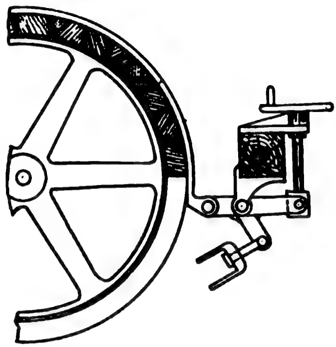


FIG. 293.—Brake wheel.

To find the gradient at which a self-acting incline will work, the following method can be employed:—

Let W = weight of full train of tubs.
w = ,, empty ,,
w_1 = ,, rope or chain.
F = friction of full train.
f = ,, empty train.
f_1 = ,, rope, rollers, etc.
I = tangent of angle of inclination.

The weight of the full load must be able to overcome that of the empty load plus the friction of both loads, and of the rope, drums, rollers, and accessories;

that is, $(W \times I) - F$ must be greater than $(W \times I) + f$, and $(W - w) I$ greater than $F + f$. From this reasoning we can establish the formula—*

$$(W - w - w_1) \times I = (F + f + f_1) \quad \therefore I = \frac{F + f + f_1}{W - w - w_1}.$$

The angle of inclination thus obtained would be that at which the pull of the full train would exactly balance the resistances on the incline, so that it would require a greater angle of inclination to enable the system to work easily.

Example.—Find the angle of inclination for a self-acting incline to act properly if the trains consist of 10 tubs, the gross load of each full tub being 16 cwts. and each empty tub weighing 5 cwts. The weight of the rope is to be taken at 5 lbs. per fathom, the length of the incline 150 fms., and the weight of drums at 8 cwts. The friction of the tubs may be taken at $\frac{1}{70}$, and that of the rope and drums at $\frac{1}{20}$.

$$\text{Then } I = \frac{\left(\frac{10 \times 16 \times 112}{70}\right) + \left(\frac{10 \times 5 \times 112}{70}\right) + \frac{(8 \times 112) + (150 \times 5)}{20}}{(10 \times 16 \times 112) - (10 \times 5 \times 112) - (8 \times 112) + (150 \times 5)}$$

$$= \frac{256 + 80 + 82{\cdot}3}{17{,}920 - 5600 - 1646} = \frac{418{\cdot}3}{10{,}674} = \frac{1}{25{\cdot}5}, \text{ or 1 in } 25{\cdot}5.$$

To make the incline work easily, it should be 20 per cent. steeper, *e.g.*, 1 in 20.

Carriage Inclines.—When the inclination exceeds 45° a carriage is employed to enable the tubs to assume a horizontal position.

Fig. 294 shows the construction of such a carriage, which may be made to hold two, four, or six tubs. These carriages are much used on the Continent, where the measures are greatly inclined. They are also used in England and in the oil-shale and coal-mines of the Lothians in Scotland. On the Continent such inclines are often worked with a balance weight with a single road, so arranged that the balance weight passes under the carriage at the point of meeting.

Blocks.—At the top of all inclines means must be adopted to prevent the loaded tubs from running prematurely down the incline. This is usually accomplished by means of blocks or stops. A common form of stop used is simply two blocks of wood working on pivots, one of which crosses the rail for the tub wheel to rest against, and abuts on the other, which is placed at right angles to it.

'Staple' or 'Blind' Pits.—It often happens that the coal is lowered from one seam to another by means of 'blind' or 'staple' pits instead of by an incline. These pits are not so expensive to make as an incline, and if properly fitted and the depth is not great a large quantity of coal can be lowered by their means. The arrangements at the top of these pits are shown in figs. 295, 296. The pits are usually fitted with two cages like an ordinary winding shaft, the cage with the full tub being able to outbalance and raise the cage with the empty tub. Sometimes one cage only is used along with a balance weight, but this is not such a good arrangement. In working, these pits ought to be carefully fenced to prevent tubs from

* It must be remembered that in the above formula I does not represent the angle of inclination, but the tangent of that angle. The former can be found from the latter by consulting trigonometrical tables.

being pushed into them when the cage is not in position. Many accidents have taken place through lack of this precaution.

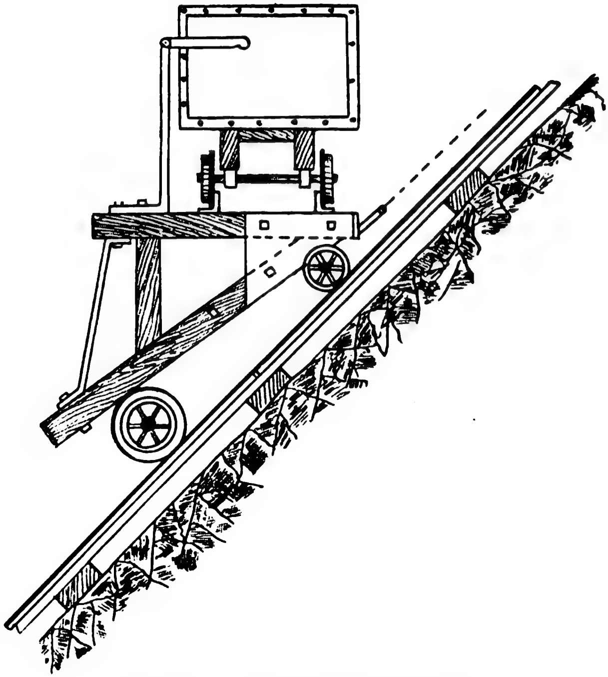


FIG. 294.—Carriage for tub on inclines.

Haulage by Stationary Engines.—We now come to the different systems of haulage by stationary engines, which play such important parts in the successful working of modern collieries. These systems can be divided into—

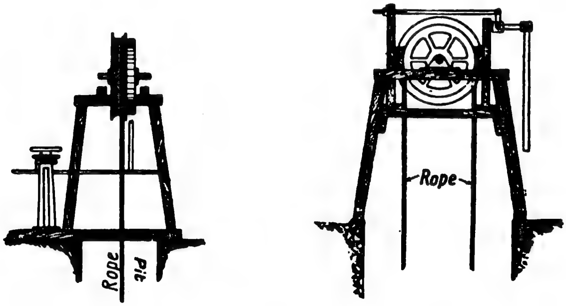
(1) Direct haulage or single rope haulage.
(2) Main and tail rope haulage.
(3) Endless chain haulage.
(4) Endless rope haulage.

The workings of a colliery may be so distributed that two or more of the above systems of haulage can be applied in different sections or localities.

Direct Rope Haulage.—When the inclination is sufficiently great for the train of tubs to run down of their own accord, and to overcome the friction of the rope to which they are attached, and of tubs, drum, etc., a single rope may be used with advantage. An engine with a single drum is employed, the latter being so arranged that it

can be thrown out of gear and run loose on the shaft. The engine draws the full train of tubs against the inclination to the pit-bottom, while the empty train runs in-bye and drags the rope after it, the drum being out of gear and running loose on the engine shaft.

To work this system successfully the inclination should be not less than 1 in 26. For single rope haulage the engine is best placed at



FIGS. 295, 296.—Arrangement of brake wheel for staple pit.

the pit-bottom, as this gives the engineman the advantage of having everything in view, and mistakes are not so apt to be made by taking the tubs too far when the train is approaching the bottom. The main advantage of this system is that it is cheap, so far as first cost is concerned, a single road and one rope only being required, while any number of branches can be easily worked by the same engine.

Main and Tail Rope Haulage.—When the inclination is insufficient or irregular, and the empty train unable to run in-bye by its own

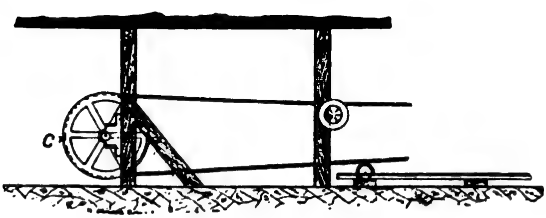


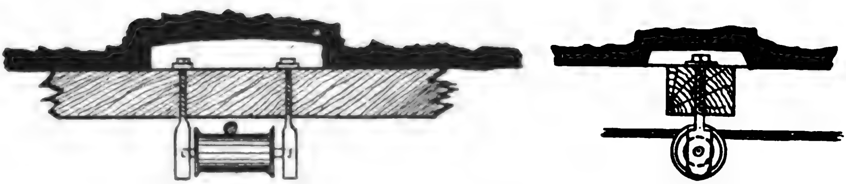
FIG. 297.—Main and tail rope haulage.

weight, it is necessary to use another rope called a 'tail' rope to draw in the empty tubs. The tail rope is wound on a drum on the same shaft as the main rope drum, and passes along the side or roof of the roadway, till it reaches the far end of the road, where it passes round a wheel *c* placed either vertically or horizontally (fig. 297), and comes on to the main roadway, where it is attached to the end

of the full train of tubs, the main rope being attached to the front of the latter. The tail rope, therefore, requires to be twice the length of the road.

Working of System.—Suppose an empty train of tubs to be standing at the pit-bottom ready to be hauled in to the workings, the drum with main rope will be thrown out of gear, and allowed to run loose on the driving shaft, while that on which the tail rope is wound will be in gear. The engine will haul the empty train in-bye with the tail rope attached to the front of it and the main rope attached to the back. When the loaded train has to be hauled out-bye these conditions will be reversed, the tail rope drum will be thrown out of gear and the main rope thrown in gear, the tail rope being now attached to the back of the train and the main rope to the front. The tail rope is usually a good deal lighter than the main rope, as it is only required to haul in the empty tubs. It will be seen from the above that this system of haulage requires a length of rope three times the length of road, the tail rope requiring to be twice the length and the main rope equal to it, so that when both ropes are connected to the train of tubs it practically becomes an endless rope working on a single road.

Guiding the Ropes.—As only a single road is used, the tail rope is guided either along the side of the road or along the roof; whichever



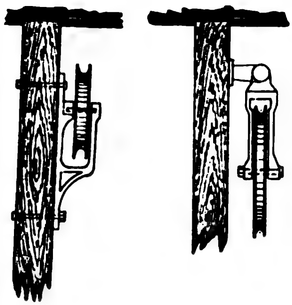
FIGS. 298, 299.—Guides.

method is adopted the pulleys ought to be set so as to run freely, otherwise there is great wear and tear on the ropes. When the tail rope is guided overhead, the pulleys are often fixed as shown in figs. 298, 299. A pulley or roller is hung by means of two hangers, about 6 in. × 3 in. × $\frac{5}{8}$ in. tapering to $\frac{5}{8}$ in. diameter at the top, where it is fixed to a bearer by means of a nut and washer. For guiding the rope along the side of the road the arrangement of pulley shown in fig. 300 may be adopted. If there are curves on the road, swinging pulleys (fig. 301) should be used, which will enable the rope to take the line best suited to it. Fig. 302 shows the general arrangement of road with pulleys for main and tail ropes.

Detaching the Rope.—When the train of tubs is near the shaft the rope is detached, and the hauling engine brought to a standstill. The rope may be disconnected either by hand or automatically. When this is done by hand a common shackle may be used (fig. 303),

the connecting bolt having an eye for a small pin to pass through to prevent it from working out.

More frequently an automatic 'knock-off' is used, as shown in fig.



FIGS. 300, 301.—Guides.

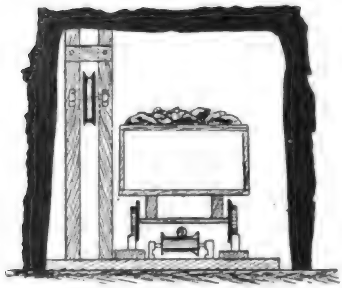


FIG. 302.—General arrangement of roads for main or tail rope.

304. Other 'knock-off' arrangements are illustrated in figs. 305, 306.

A somewhat different arrangement for automatically detaching the rope is shown in figs. 307, 308. This apparatus, which is often

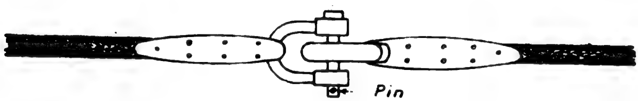


FIG. 303.—Shackle.

termed a 'monkey,' is placed on the front of the train of tubs, and consists of a crank *a*, working on a standard *b*, which is fixed to the tub by a fork arrangement, or special clamp fitted to the tub. To

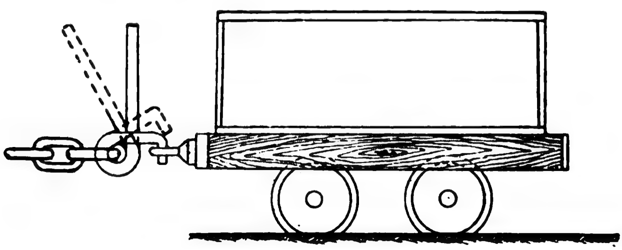
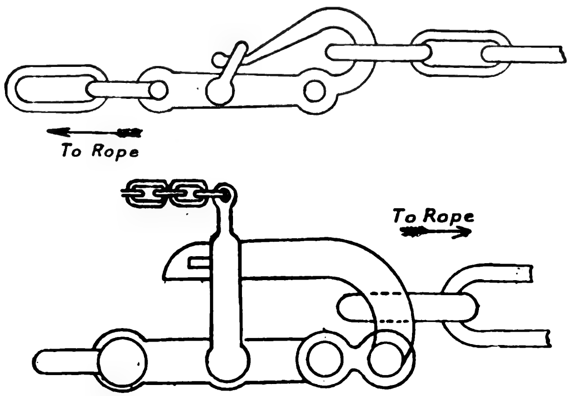


FIG. 304.—Automatic knock-off arrangement.

the end of the crank is a chain *c*, with a pin at the end for fixing the rope. At the place where it is desired to detach the rope, a beam is fixed across the road sufficiently low to strike the upright

lever of the crank, thus raising the chain with the pin and releasing the rope.

Working Branch Roads.—Branches from the main road are worked by means of separate ropes, which pass round a wheel at the extreme end of each branch, and extend to the junction with the main road. The working and connecting of these branch ropes may be done in three different ways. In two of these the connection is made when the empty train is brought to the junction of the branch road, while in the third method the connection is made while the empty train is standing at the shaft. In the first method, if an empty train requires to be taken into branch *a* (fig. 309), the end *b* of the main line tail rope is disconnected from the train, and the end *b′* of the branch rope is attached in its place. The main line tail rope is also cut at

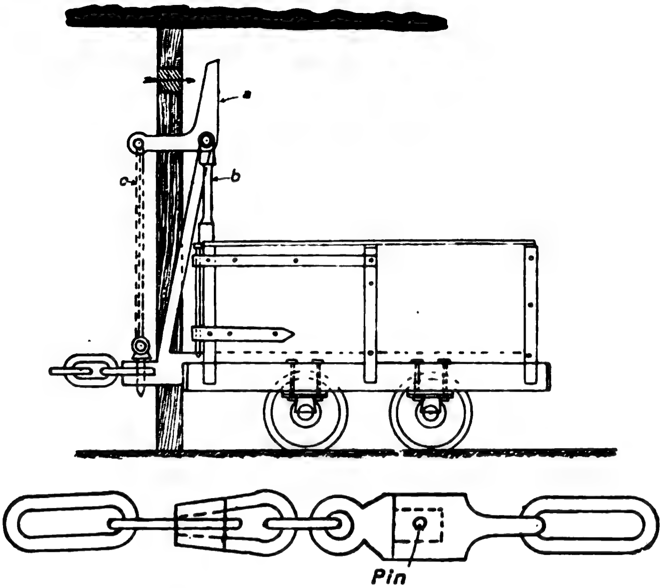


FIGS. 305, 306.—Other knock-off contrivances.

a, and the other end of the branch rope, *a′*, connected to it; the tubs are then drawn into the branch road. In the second method (fig. 310) the main line tail rope is disconnected at *b*, and the end *a* of the branch rope connected to the tubs; the engine now draws the rope forward a short distance, and the other end of the rope is connected to *c*, the train being then ready for hauling in. In the third system the ropes are so arranged that when the full train on the main road reaches the shaft, the two disconnecting points *y y* (fig. 311) are just opposite the branch *c*. If a supply of tubs is wanted in this branch, the rope is 'cut' at *y y*, and the ends *x x* of the branch rope are connected instead, before the train of tubs leaves the pit-bottom; a run can then be made into the branch without stopping at the junction, which makes the work both simpler and more expeditious.

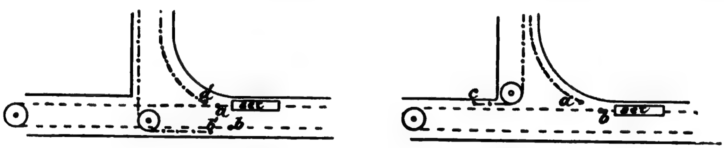
The main and tail rope system of haulage is extensively used in some districts, notably in the North of England, where at one time

it was employed to the exclusion of every other method. The speed at which the rope travels is high, varying from six to twelve or fifteen miles per hour. To ensure success with this system, the roads should be as straight as possible, and well laid with heavy rails. Where



FIGS. 307, 308.—The 'Monkey' knock-off.

there are curves on the road, the rope should be carefully guided with bevel pulleys, and 'check' rails used to keep the tubs from becoming derailed. A modification of this system is sometimes adopted, in which an endless rope is worked in opposite directions as



FIGS. 309, 310.—Working branches.

required, but this method is not at all to be recommended, and should only be used under exceptional circumstances.

The advantages of the main and tail rope method of haulage are that only a single road is required, the first cost being consequently less than with endless rope haulage with double road; roads with

irregular inclinations and curves can be worked easily, although the system is best adapted for straight roads with regular inclinations, and branch roads can be worked with facility. But against these advantages must be placed the following disadvantages in the working of the system :—

(1) The high rate of speed at which the rope travels causes fast wear of the rope, and much damage if the tubs get derailed.
(2) The irregular delivery of tubs at the pit-bottom.
(3) Larger engines are required to perform a given amount of work, and a larger amount of steam power required than with endless rope haulage.
(4) The outlay for engine and ropes is greater, as the rope requires to be three times the length of road.
(5) Greater danger for men and horses travelling on the roads, unless a separate travelling way be provided.

The engines for working the main and tail rope should have two cylinders, and be geared to the required proportions.

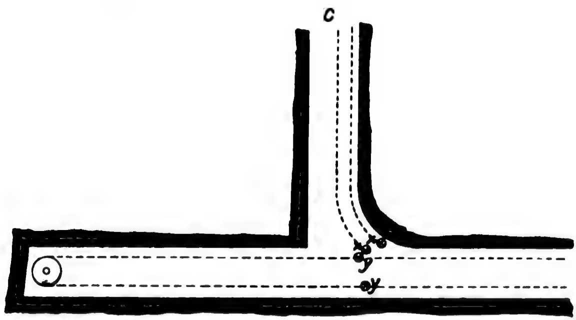


FIG. 311.—Working branch.

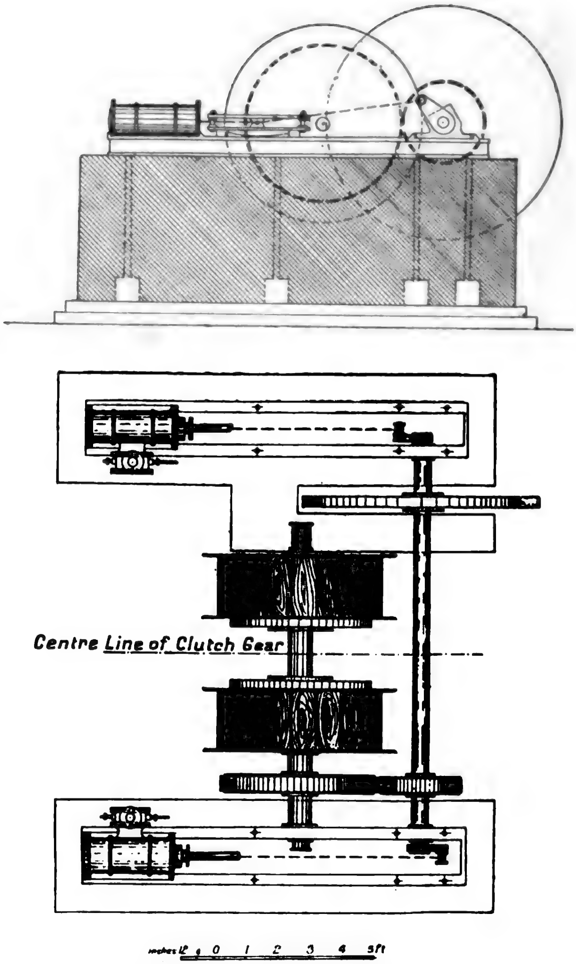
Figs. 312, 313 show the plan and elevation of a good arrangement of engines for this system, both drums being placed on one driving shaft. The engines, as in the direct rope system (and for the same reasons), are better placed underground.

Endless Chain.—This system of haulage is in all respects similar to the endless rope system to be described shortly, although the speed is somewhat less and the tubs are attached at closer intervals. It differs from the main and tail rope system, as a double road is required, and the tubs are attached singly instead of being attached in trains; the speed is also much lower, the endless chain being driven at a rate of one and a half to three miles per hour.

The endless chain may be worked either by an engine and wheel above ground, which is the most common arrangement, or by an engine and gearing placed underground.

Chain Wheel.—For driving the chain, a wheel with a broad rim to allow the chain to make two or three laps on it is often used. Sometimes a wheel, such as that shown in fig. 314, is used. It has a broad

rim, and a number of 'feet' or 'studs' are screwed into the face which grip the chain and prevent it from slipping. The feet are so arranged that the distance between them is exactly equal to the



FIGS. 312, 313.—Arrangement of haulage engines for main and tail rope.

length of a link. In Briart's wheel for endless chain a series of V-shaped grips or feet of steel are screwed into the rim, as shown in fig. 315. As the links of the chain stretch by wear, the feet can be

unscrewed and fixed in a new position to suit the altered lengths of the links.

Sometimes a series of steel blocks are fitted into the circumference of the wheel, and the chain coiled round two or three times to give the required grip. The blocks are secured to the wheel by means of counter-sunk bolts, the part on which the chain works being octagonal in shape, which serves to prevent the chain from slipping.

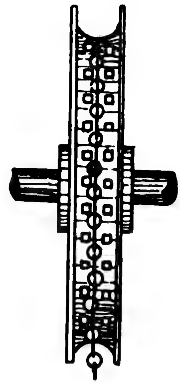


FIG. 314.—Chain wheel.

Lengthening the Chain.—It is often difficult to keep the chain tight, owing to the continuous lengthening due to wear, and a common practice is to allow it to stretch until it becomes necessary to remove a portion to tighten it. It is, however, better to use a 'tightening' carriage similar to that used with the endless rope system.

Working Branch Roads.—Branch roads off the main line can be easily worked with endless chains, and this is one of the chief recommendations of the system. The working of the branches will be understood from fig. 316. In this method three separate chains are used, one for the branch and two on the main road. These chains are worked by a series of bevel-geared pulleys working at right angles to each other. On the shaft to which the horizontal pulleys are attached, a driving wheel is fixed round which the chain passes. The whole of these wheels and gearing are placed below the level of the roadway and boarded over. At a short distance on either side of the branch the tubs are detached from, and again attached to, the chain by an automatic arrangement which is shown in fig. 316. Travelling in the out-bye direction the chain is gradually raised to near the roof by a large pulley working on a shaft (fig. 316); a small roller pulley guides the chains to this larger pulley, placed horizontally. The road for the loaded tubs is given an up gradient towards the shaft, and the empty road has a dip in the opposite direction. These gradients commence a short distance on each side of the branch. By the rope rising towards the roof and the road dipping the chain is raised out of the gripping fork; the tub being released runs over the part where gearing is placed, and re-connects itself to the chain again automatically. Exactly the same arrangement is required on the empty or in-going chain.

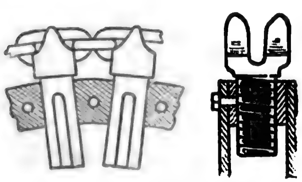


FIG. 315.—Briart's wheel.

Branch roads may also be worked by a series of wheels fitted on to

a driving shaft placed vertically, the branch wheels being arranged so as to work with clutches to enable them to be thrown in and out of gear as required. The main driving wheel is keyed tightly to the shaft. This arrangement allows of a branch chain being arrested if anything goes wrong, or if there are not sufficient tubs to keep it constantly moving.

Working Curves.—If a curve is one of short radius, a pair of large wheels are placed at the bend round which the chain passes (fig. 317). The tubs are detached automatically before coming to the curve in the manner already described for working branch roads. When they

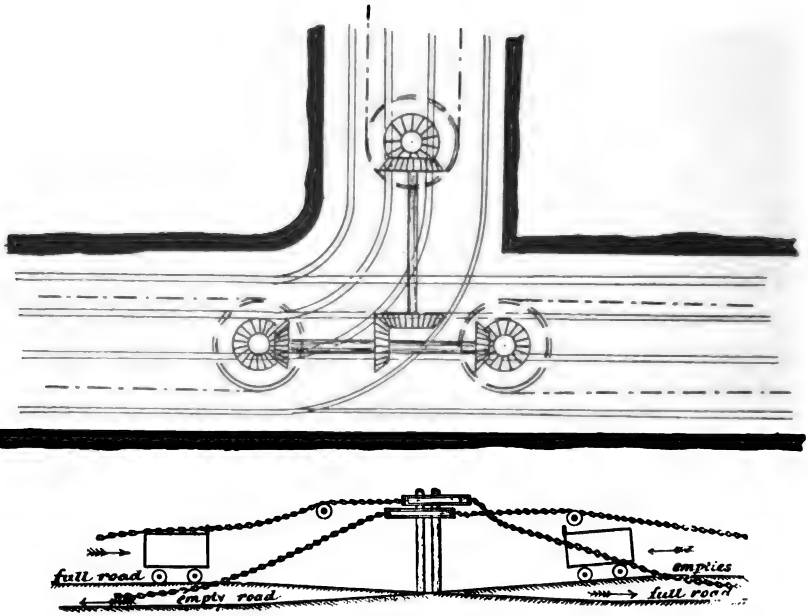


FIG. 316.—Working branches.

pass round the bend they again attach themselves automatically to the chain.

Method of attaching Tubs to Chains.—This is usually done by fixing to one end of the tub an iron fork, into which the chain drops. When wood tubs are employed, a fork is used, being fixed by means of two nuts and bolts. With iron tubs the grip is a part of the tub itself, and requires no fixing.

Sometimes movable grips are used to adjust the height of the chains according to the quantity of coal loaded above the level of the sides of the tubs.

The chief advantages of the endless chain system are the slow speed

at which it travels, and the small amount of wear and tear in rolling stock entailed, besides the small cost of upkeep of roadway. The principal disadvantage is the heavy weight of chain to be driven where the haulage is long.

This system is better suited for surface haulage than for underground working, many such arrangements being at work for conveying coal or other material long distances, as when a colliery is so situated that it would be impossible or inconvenient to connect it with the railway by means of a branch line.

Endless Rope.—This system, as its name signifies, consists of an endless rope travelling in a double roadway, and to which the tubs may be attached either singly at intervals along the rope or in trains. The engine for driving the rope is almost invariably placed on the surface,

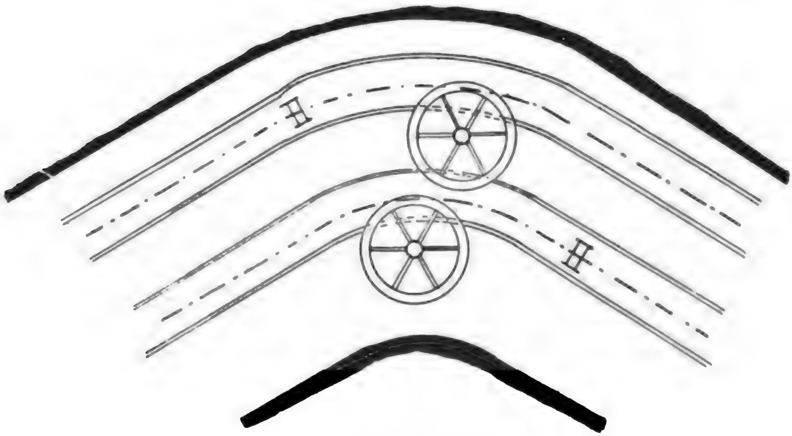


FIG. 317.—System of working curves.

and the power either conveyed direct or by a 'band' rope reaching to the pit-bottom, where it drives a main shaft from which the power is derived for other endless ropes. The general arrangement in endless rope haulage (fig. 318) is as follows:—At the mouth of the shaft are two pulleys for carrying the rope from the driving wheel into the shaft. At the pit-bottom are two other pulleys, placed vertically to receive the ropes, and immediately below these are two other pulleys, placed horizontally to enable the rope to make a right angle with the shaft, and pass into the workings to the in-bye end, where it passes round a loaded wheel *d* or an ordinary pulley placed horizontally.

In another arrangement the loaded wheel or tightening carriage is placed on the empty rope side near the pit-bottom. This is undoubtedly the best position.

Arrangement of Engine.—For endless rope haulage, where a large quantity of coal has to be drawn, it is best to employ a double cylinder

engine well geared down. With a modern haulage plant for endless rope, the two cylinders are placed horizontally, and the piston rods connected to disc cranks fitted on to the main driving shaft. On this shaft are two small geared wheels, working into two larger toothed wheels, which are keyed on to a separate shaft, on which is also fixed the driving pulley or pulleys, the number of which varies with the number of ropes to be driven. These pulleys are now usually arranged with clutch gear so that they may be rapidly arrested without bringing the engine to a standstill. As to whether the power should be conveyed through a band rope or not, there are differences of opinion. Where more than one endless rope is required to work different sections, the band rope method is to be preferred. If four different sections are worked by endless rope, and each separate rope is worked direct from the surface, this would necessitate four double lengths of rope being conveyed down the shaft, which, owing to the complications that might arise in the event of any of the ropes breaking or getting entangled with each other, would be altogether undesirable. On the other hand, when a band rope is used only a double length of rope requires to be led down the shaft. It must not be forgotten, however, that as the whole of the power must be conveyed through the band rope, this would require to be a good deal heavier, and therefore more expensive than the haulage ropes worked from it, while there would also be the increased expenditure for the clutch arrangement at the pit-bottom. With the band rope system, any number of ropes can be worked, according to the power available. They are under the direct control of the attendants at the shaft bottom, who can stop any of them without signalling to the surface, which is a great advantage, as one section may not have sufficient coal to keep the endless rope

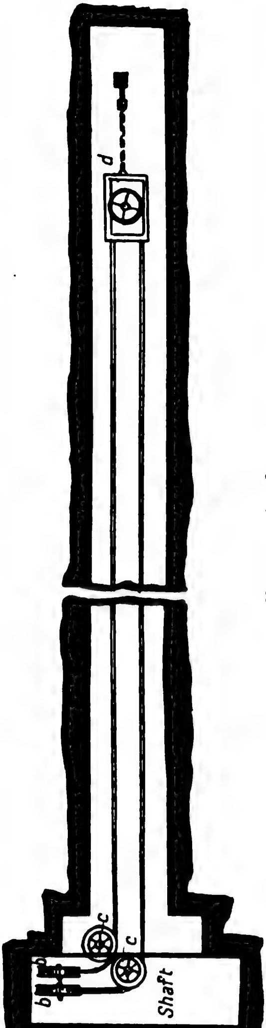
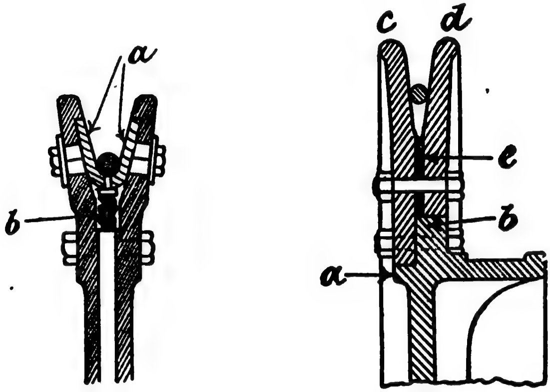


FIG. 318.—Endless rope haulage arrangement.

constantly at work, and such rope can therefore be thrown out of gear, while the other sections go on as usual.

For a very deep shaft the cost of the band rope becomes, on the other hand, a serious item, although even then it would be less expensive than taking all the ropes up to the surface. Everything considered, the band rope system of conveying the necessary power is much to be preferred.

Driving Pulleys.—The rope is usually actuated by a clip pulley. There is a considerable number of different types of pulley in use. The Barraclough pulley is a well-known device for endless rope haulage. Round the periphery of the wheel and opposite to each other are fixed a number of short taper clips fitted to receive two sliding jaws *a a* (fig. 319), upon which the rope works. These jaws



FIGS. 319, 320.—Barraclough's pulley.

rest on springs *b*, so that when the rope comes on to them, the weight forces them down on the springs, and narrows the opening between them, thus giving the necessary grip to the rope. At the point where the rope leaves the pulley the springs give assistance in releasing it.

Another pulley, much used in Scotland, is shown in fig. 320. The rim consists of two segments *c* and *d*, bolted together. Between these segments is placed a layer of wood *e* on which the rope works. The opening on the rim of the pulley being V-shaped, the rope obtains the necessary grip by wedging itself at the bottom of the opening.

Owing to the low coefficient of friction between iron and iron, and also to the wear of the rope in unlined pulleys, it has become customary to pad them with softer material than iron, to increase the gripping power and so increase the life of the rope. Segments of hard wood are used, but require frequent renewal, and soon lose their gripping power. Segments of india-rubber have been tried with good

effects, and last a considerable time. Possibly the best substance for this purpose is well-seasoned leather driven into the rim of the wheel in segments *a a* (fig. 321), and turned true. This is said to give a good gripping surface that will last from two to three years. When a large amount of power requires to be transmitted, grooved pulleys and counter-pulleys or lacing wheels are sometimes used to impart the necessary friction to the rope. Both pulleys have from four to six grooves in the rim, the driving wheel *a* having one more than the guide or counter-pulley *b*, which is set immediately in line with the other (figs. 322, 323).

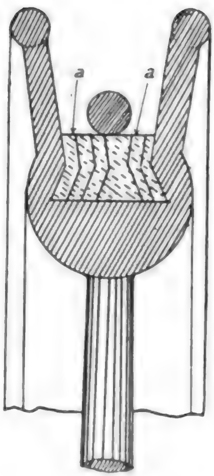
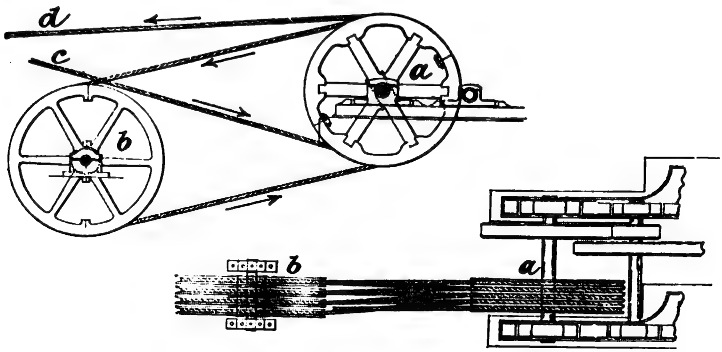


FIG. 321.—Method of gripping rope.

Taking up Slack Rope.—No matter how perfect the driving pulley on the surface may be, some arrangement for taking up slackness will require to be adopted. The usual plan is to have the rope passing round a tightening pulley.

There are various ways of arranging this pulley. Sometimes it is fixed as shown in fig. 324, and fitted with a long screw and nut, which can be adjusted as required. But this system of tightening is not at all to be recommended. What is required is some arrangement that will automatically adapt itself to the varying load and wear of the rope. To accomplish this, a wheel is usually mounted on a tension carriage to which is attached, by means of a



FIGS. 322, 323.—Grooved pulleys.

chain passing over a pulley, a weight (see fig. 325) working in a small pit below the level of the road. Sometimes the tension-carriage is attached to a loaded tub, working on an inclined plane. The

weight required can easily be found when the rope is set to work. The tension pulley is usually placed as near the pit-bottom as possible, although it may also be placed at the further end of the road. It is, however, often very inconvenient to have it in the latter position.

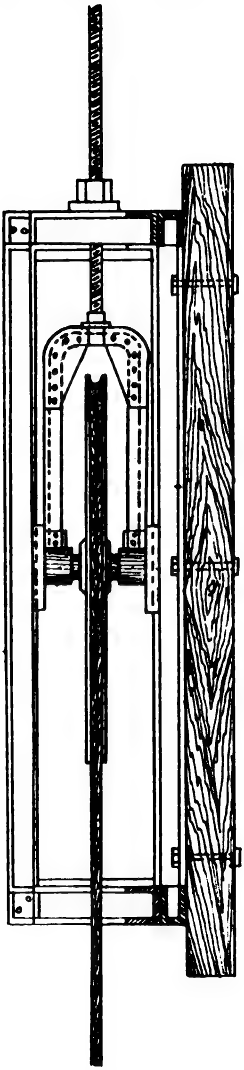


FIG. 324.—Fixed tightening pulley.

Speed of Rope.—With endless rope haulage, the speed at which the rope travels may vary from $1\frac{1}{2}$ to four or five miles per hour, but the best results are obtained when the rope is travelling at a speed of from two to three miles per hour.

Working of System.—The system can be applied either: (1) with the rope travelling over the tubs; or (2) with the rope moving beneath them. The tubs are also attached to the rope in two different ways: (*a*) singly at specified distances apart, by means of 'clips' or 'jiggers'; (*b*) in sets or trains by means of a 'clip-bogie' and gripper.

The first method is undoubtedly the better of the two, as by this system the supply of coal at the pit-bottom is regular, and the load on the engine more evenly distributed, while no attendants are required except at each end of the road or at branches.

Some, however, prefer the 'clip-bogie system,' as being safer, and better adapted for working a number of branch roads. It is somewhat more expensive than the first-named method. With either system a double road is generally employed, although sometimes a single road is used with pass-byes at intervals. When such is the case, the tubs are run in trains. It is often urged as a defect of the endless rope system, that it requires a double road, the construction of which is often difficult and expensive, especially where there is a bad roof. But a double road is not at all essential for the successful working of an endless rope system of haulage.

At the Palace Colliery and Bent Colliery, Hamilton, systems of endless rope haulage have been successfully at work for years, in

both cases arranged on the single road principle. In the first the rope passes under the tubs, while in the second it is overhead, the latter being probably the more successful method.

At a short distance from the shaft are set two parallel single roads, from different sides of the pit-bottom. These two roads are carried for the required distance, and are connected by a cross drift into which the rope passes by means of two wheels. These two parallel roads may be as wide apart as required. The empty rope and empty tubs leave the pit-bottom at one side, while the full rope and loaded tubs arrive on the opposite side. This system has a great advantage over that where the empty and loaded tubs alike require

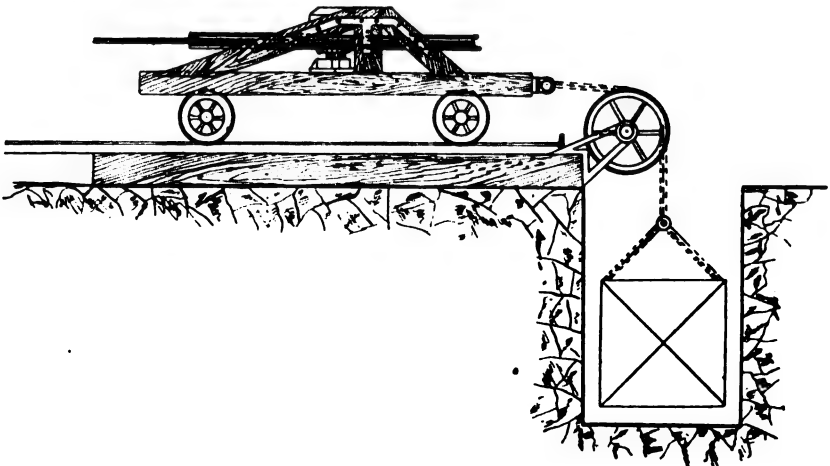


FIG. 325.—Tension carriage.

to be handled at the same side of the shaft. The rope can be easily extended into the workings.

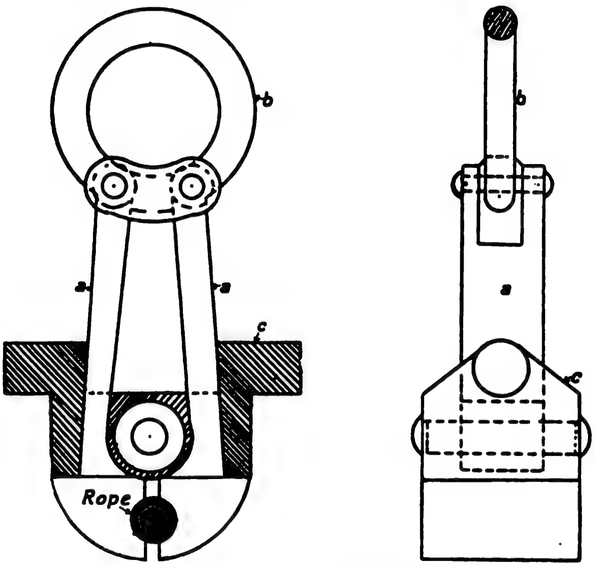
At the Bent Colliery much the same system was adopted, but the two roads, instead of being driven parallel, were taken along a more circuitous route, practically to follow the faces, which enabled the tubs to be brought on to the rope with very little secondary haulage. The empty tubs can be taken off, and full tubs attached at any desired part of the road, and if the empty tub is not required, or is not taken off the rope in the workings, it will return again to the pit-bottom on the side opposite that from which it left.

Clips or Jiggers.—When the tubs are attached singly to the rope, a clip or jigger is used. A great many different sorts of clips are in use in different districts, each having its own special merits.

A clip used in Scotland a good deal for under-rope haulage is shown in figs. 326, 327. It consists of a pair of jaws *a a*, which are attached to

a large link b, the latter being in turn attached to the tubs. Over the two jaws a a passes a cone or thimble c, which, on being driven hard down, presses the two jaws together and grips the rope, which is caught in a groove at the bottom. This clip is easily attached to and detached from the rope, and is very simple in construction, but it is rather bad for the rope, and is best suited for comparatively level roads. To automatically detach the tubs from the rope, when this clip is used, the arrangement shown in figs. 328, 329 is adopted.

At the in-bye and out-bye ends of the road a small inclined plane, with an opening in the centre for the rope to travel in, is made in the

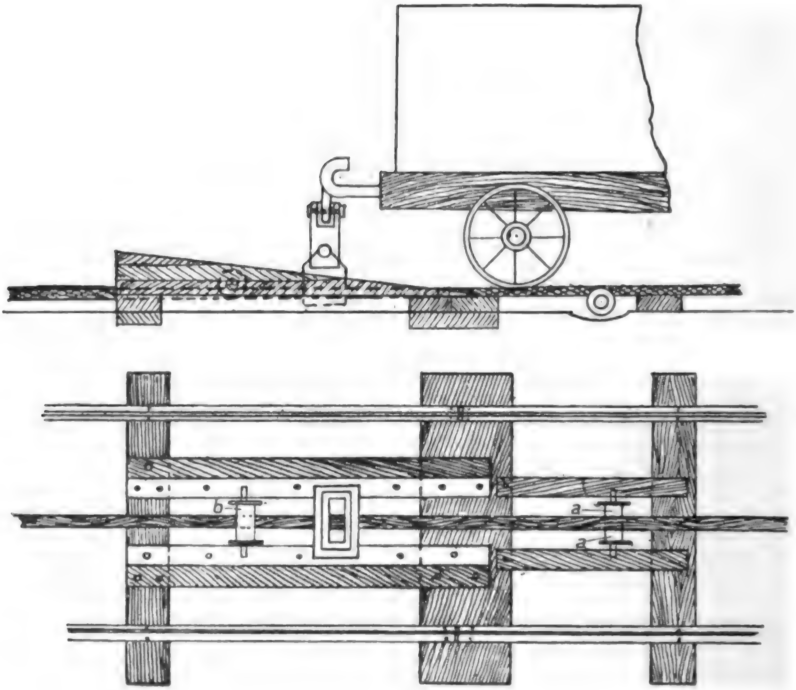


FIGS. 326, 327.—Clip or jigger for endless rope.

centre of the roadway. This opening is just sufficient to admit the rope and the bottom of the clip, the cone or thimble edges being caught on the inclined plane, which rises until it is sufficiently high to raise the cone, and allow the jaws of the clip to open and so release the rope. To prevent the latter from rising, two pulleys are fixed at each end of the inclined plane, the rope passing over the pulley a and under the pulley b. When the clip has been in use for some time it gets worn and fails to grip the rope properly, so that if the thimble becomes at all slack the tub may come to a standstill, and the rope continue to run through freely. This causes much undue wear, but can be avoided by using steel lining pieces which can be fitted into the jaws of the clip and renewed when required. This

clip has been known to work successfully on roads with a gradient of 1 in 7.

Another clip employed is that known as Fisher's patent, which is illustrated in figs. 330, 331. As will be seen, it also consists of two jaws *a* and *b*, the part *a* working on a pin *h* at the bottom of the clip, which enables it to fold over the rope. The two jaws are held in position by a sliding thimble *c*, as described above. The clip can be auto-



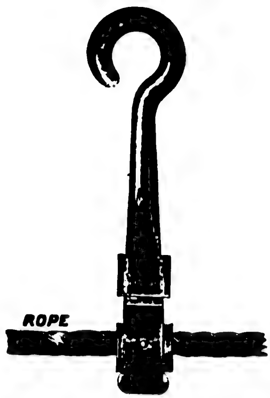
FIGS. 328, 329.—Automatic detaching arrangements.

matically detached in a similar manner to that described for the first clip. A different arrangement of jigger and automatic detachment is shown in fig. 332.* The detaching apparatus is rather more complicated than that already described. It consists of a horizontal axle *a*, carrying two arms *b b′*, of equal length. One of the arms, *b′*, has a circular disc which passes up through the centre of a split rail, and projects about 5½ in. above it. The tub wheel on passing depresses this arm 5½ in. and at the same time raises the other arm an equal distance. This other arm *b* carries a steep cone roller *c*,

* *Trans. Inst. Min. Eng.*, vol. viii. p. 377.

which is raised with it, and thus raises the rope and slips the cone down clear of the jigger altogether.

Other clips, such as Smallman's, Hanson's, and Humble's, are also used for attaching the tubs for under-rope haulage. The Smallman clip is a very good one for under-rope haulage, and can be used on



FIGS. 330, 331.—Fisher's clips.

roads of gradients 1 in 4 or 1 in 3. With overhead haulage a different sort of clip is required. The simplest kind for this purpose is that shown in fig. 333. It is placed either in the centre or side

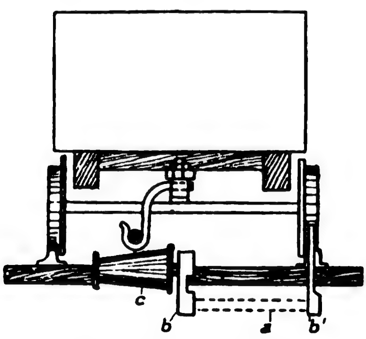


FIG. 332.—Automatic detacher.

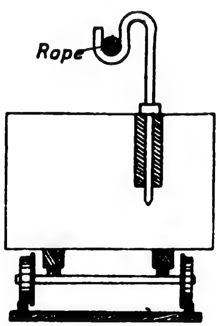


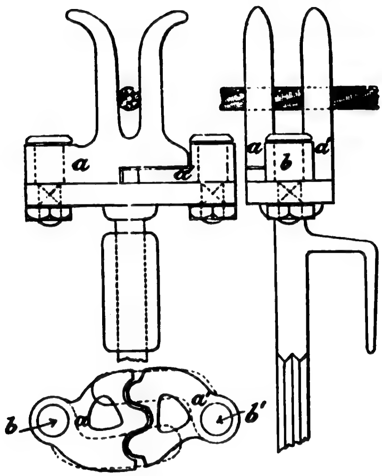
FIG. 333.—Rope clip.

of the tub, and on gripping the rope it gives it a slight twist, which gives it the necessary hold to carry the tub along. Sometimes two of these jiggers are used, one at the back and one in front of the tub.

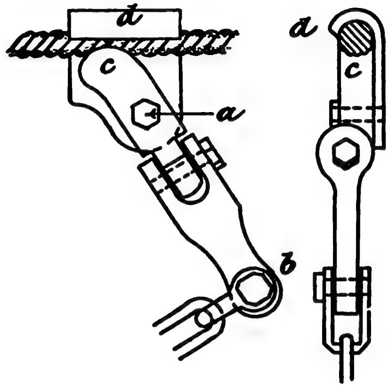
Rutherford and Thomson's clip, as shown in figs. 334, 335, is also

used for overhead rope haulage. This clip is an application of the principle of the toggle joint, and its grip on the rope is equally firm in whatever direction the load may act. It is equally well adapted for steep inclinations as for roadways of varying gradients. With this clip curves and junctions can be worked automatically on the gravity principle in the same way as in endless chain haulage.

Ward and Lloyd's clip,* as described by Mr H. W. Hughes, seems to be very simple and efficient for overhead rope attachment. It consists of a hinged lever, to the bottom of which is attached the chain fastened to the tubs (figs. 336, 337). The lever works about



FIGS. 334, 335.—Rutherford and Thomson's clip.



FIGS. 336, 337.—Ward and Lloyd's clip.

a pivot *a*, and when the weight comes on the end *b*, the rope is gripped between the top end *c* and the curved plate *d*. The lever is hinged, which allows the clip to fall into the guide pulleys when passing round curves. This grip is best suited for roads having a regular inclination, otherwise two clips will have to be used for each tub.

Running the Tubs in Sets.—When the tubes are attached to the rope in trains or sets, a bogie with a gripper or shears is used.

Fig. 338 illustrates a form of bogie clip which is much used for this purpose. The gripper swings on a pivot which enables it to pass round level pulleys at curves; it is actuated by a screw and handle in much the same way as the brake screw of a railway van. On inclined roads these clip bogies are usually loaded with scrap metal in front to keep them from 'rearing' and getting derailed.

Catch Blocks.—On inclined roads it is often the custom to have

* *Text Book of Coal Mining*, Fifth Edition, p. 271.

catch blocks or runaway catch points to arrest the tubs should they happen to become detached from the rope. A simple form of catch block is shown in fig. 339. It consists of a piece of wood fixed on a spindle, the latter not being in the centre. When the tub passes over it the wheel axle depresses it to a horizontal position, but as soon

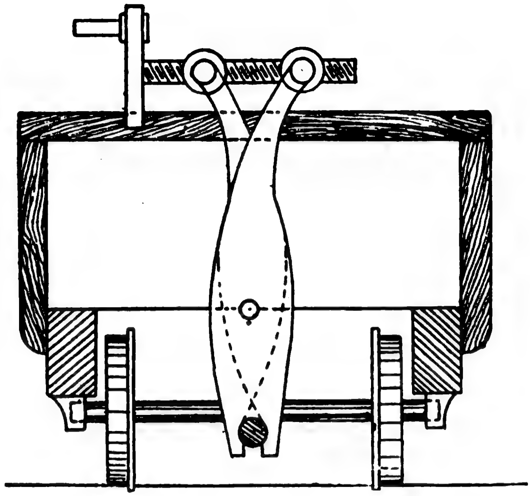


FIG. 338.—Bogie clip.

as the tub is passed it assumes its inclined position again, and will be ready to grip the tub should it run back. Runaway points fixed to the rail and having a strong spring are used for the same purpose.

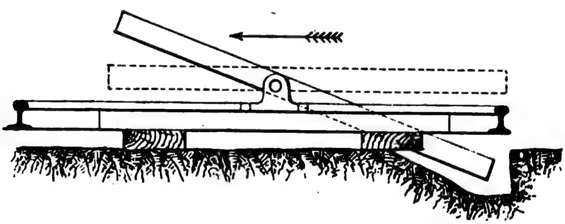


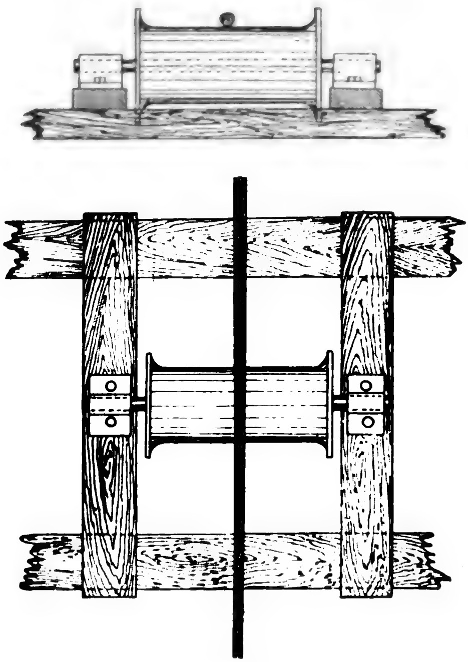
FIG. 339.—Catch block.

The tub on passing opens the points, which close of themselves when it has passed through.

To get the best results from endless rope haulage, the rope should be carefully guided by means of rollers to prevent it from rubbing on the floor and to keep the sleepers from getting cut through. These rollers may be made either of wood or iron, and properly fixed so as

to run freely, a point which is very often neglected, giving rise to much unnecessary friction and wear of the rope. Figs. 340, 341 show the method of fixing these pulleys on haulage roads. When the rope has to be taken round curves it must be guided by means of bevel pulleys, as shown in figs. 342, 343.

Endless Rope on Inclined Roads.—It has often been urged that endless rope haulage is only suitable for flat or comparatively flat workings; and while no doubt the best results are obtained under



FIGS. 340, 341.—Roller pulleys.

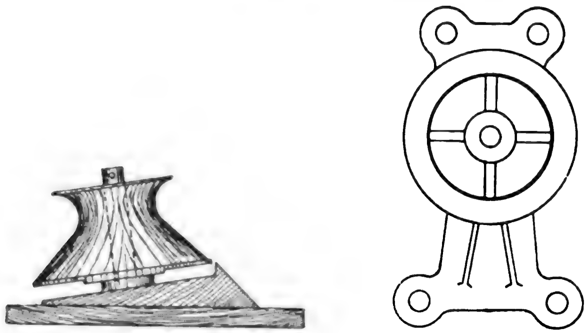
such conditions, still the system can be successfully worked at inclinations of 30° or more.

At Moston Colliery, Manchester,* there is a good example of this system of haulage on steep gradients. The road in question is 2900 ft. long, the inclination varying from 26° to 39°. The rope travels at a speed of 1½ mile per hour, and runs beneath the tubs, which are attached to it by means of a stout parallel jaw-clip. The jaws of this clip are actuated by a screw and hand-wheel, and a hinged iron bar serves to connect it with the draw-bar of the tub, and prevents the clip from turning round and fastening itself under the rollers

* *Trans. Min. Inst. Scot.*, vol. ix. p. 120.

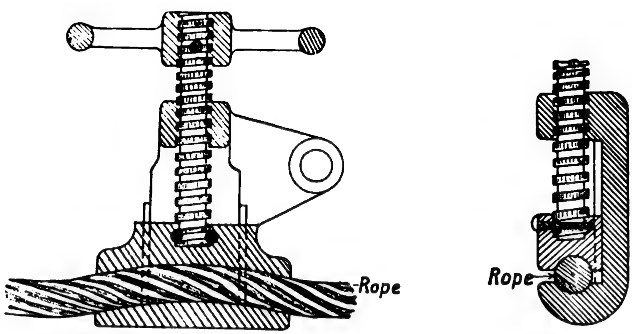
between the rails. It also assists the tub to keep the rails while being attached to the rope, at the time of leaving the level landings.

The clip used is shown in figs. 344, 345. The jaws are made to a radius of 3½ ft., so that when the screw is tightened the clip has a firm grip of the rope in at least three places, at each end and in



Figs. 342, 343.—Bevel pulleys.

the centre. Ordinary clips were found to be of no use in this case, and hence the adoption of this special type. The full tubs are attached singly, and the empty ones sent down in pairs; both the attaching and detaching being accomplished at all points without stopping or interfering with the speed of the rope.



Figs. 344, 345.—Endless rope clip used at Moston Colliery.

There are several intermediate landings at which the full tubs are attached and the empty ones detached. Fig. 346 shows the method of working these landings. Sufficient roof is taken down to allow one full or empty tub to stand on the fixed landing A B. There is a movable hinged landing A C, controlled by a balance weight in such a position that when it is lowered the empty tubs will run

on to this platform instead of going further down the incline. When the tubs have arrived the clip is unfastened, and they are run on to the fixed landing A B, the movable landing A C being immediately pulled up by the balance weight into the position shown in the figure until it is required again. When a full tub is about to be clipped on to the rope, the movable landing A C is lowered, and the tub brought on to it, and without stopping the rope it is clipped and the tub started up the incline. Two boys are stationed at each of these stations to attend to the tubs. The distance between the landings varies from 100 to 140 yards. The power

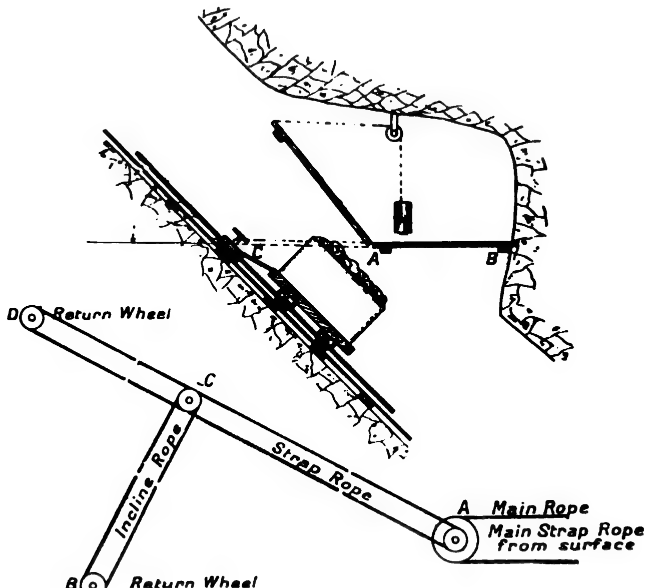


FIG. 346.—Method of working landings at Moston Colliery.

to work this and other ropes is derived from an engine at the surface, being transmitted by means of a band-rope working to a central station underground, the driving engine being compound, with cylinders 15 and 25 in. respectively, and having a stroke of 4 ft. All the pulleys underground are inclined (fig. 347) on the rope surface to the extent of $1\frac{3}{16}$ in. in 6 in., so as to cause the $2\frac{1}{2}$ turns of the rope (which are necessary to give the required grip) to be constantly slipping in the direction of the lesser diameter.

Attached to the pulley working the incline is an ingenious brake arrangement. The brake-rim of the pulley is cast and faced with steel segments. Upon this surface four steel slipper-blocks *a a*

(fig. 348) rest, attached to the framework by means of toggle joints *b b*, each pivoted on to a plummer block *c c*.

These slipper-blocks are not set at right angles to the brake-rim, but at such an angle that, upon the wheel continuing to turn in the proper direction (shown by the arrow), the blocks are pushed off the rim; but upon the incline rope being thrown out of gear, the screws *b b* tend to take a position at right angles to the brake-rim, and in so doing wedge the blocks firmly against the rim, and prevent the wheel from turning the opposite way. The rope is thus prevented from a backward movement. To further increase the braking power of the slipper-blocks, they are also attached to the frame by light spiral springs. When the pulley is thrown out of gear, its first tendency is to revolve in a contrary direction, owing

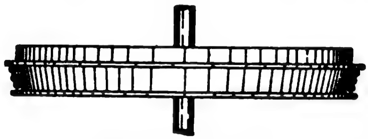


Fig. 347.—Pulley.

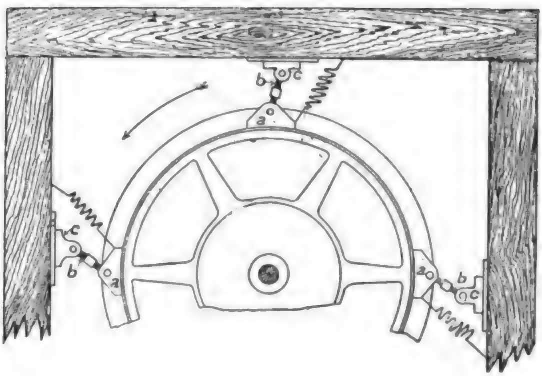


Fig. 348.—Brake arrangement.

to the weight of tubs and rope on the incline, but this movement is immediately arrested by the automatic blocks coming into action.

The incline for this haulage was laid with a double road throughout, with steel rails weighing 24 lbs. per yard, and fish-plate joints. The amount of coal drawn per shift of nine hours was 300 tons, which is excellent work on a road of this description.

Cost of Haulage.—The cost of haulage is usually stated in pence per ton per mile hauled, and will vary a great deal under different conditions, such as the inclination of the road, the number of

branches to be worked, and whether the rope can be kept continually at work during the shift or not.

The following may be taken as average costs :—

Manual haulage	1s. 6d.	to 3s.	per ton	per mile.
Horse ,,	3d.	to 6d.	,,	,,
Self-acting incline haulage	2d.	to 3d.	,,	,,
Single rope haulage	2d.	to 6d.	,,	,,
Main and tail rope haulage	1¾d.	to 2¼d.	,,	,,
Endless rope (clip) ,,	1d.	to 2d.	,,	,,
,, (bogie) ,,	2d.	to 3d.	,,	,,
Endless chain haulage	½d.	to 1½d.	,,	,,

There can be no doubt that, with a well laid out haulage plant and under suitable conditions, the endless rope system is preferable in most cases. Even in steep roads it will compare favourably with other systems of haulage. In fact, the author is of opinion that the endless-rope system is the most suitable for underground work, whether the roads are flat or steep, and can be successfully worked if the roadways are properly laid out for such haulage.

In two instances,* with single rope and endless rope systems, one of the roads dipping 1 in 3 and the other 1 in 4, and each being about 1200 yards long, and yielding an output of 800 tons per day, the cost per ton in the former was 5d. and in the latter 1d. per ton. The cost of ropes by the endless system was 0·37d. per ton, which could not be any lower in any other system; for although a double length of rope was required, the difference in cost is always compensated by the reduced wear and tear. There will not, naturally, be the same disparity of cost in all cases, but, as a general rule, endless rope systems will be found to be cheapest.

Advantages of Endless Rope Haulage.—While the endless rope system has the disadvantage of requiring double roads, it has many advantages to compensate for this. Amongst others are—

The small number of attendants required for a given daily output.

The slow speed, which prevents any loss on the journey, reduces to a minimum the risk of breakages, and thereby obviates the mischief consequent on such accidents, such as those which may occur when a train of tubs is travelling at a high rate of speed in the main and tail rope system.

The wear and tear on machinery, tubs, ropes, etc., is a great deal less than with other systems of haulage.

By attaching the tubs at equal distances apart, the weight of the rope is carried along with less friction on the ground and pulleys.

A regular and continuous supply of tubs is brought to the pit-bottom.

In a road with varying degrees of inclination, the whole load is distributed over the whole length of the rope, which is a great advantage. The empty tubs going in-bye also assist the engine to haul the full tubs out-bye.

Ropes last longer (on an average three to four years), while with main and tail rope the average is probably not more than one to one and a half years.

* *Trans. I. M. E.*, vol. x. p. 497.

Haulage Problems.

Question.—A train of ten tubs is ascending an incline of $4\frac{1}{2}$ in. rise per yard, each tub weighing with coal one ton. What power would be required, and what would the strain on the rope be?

$4\frac{1}{2}$ in. per yd. $=\frac{3 \times 12}{4 \cdot 5}=1$ in. in 8.

Weight of train $=22400$ lbs.

Friction taken at $\frac{1}{70}$ of the load $=\frac{22400 \times 1}{70}=320$ lbs.

Then $\frac{22400 \times 1}{8}=2800$, the force required estimated in lbs., and

$2800+320=3120$ lbs. = the total strain on rope.

Question.—Find the tension on a haulage rope with a load of 20 tons on an incline of 1 in 6.

Let W = load in lbs., H = vertical factor of rise, and L = horizontal factor of rise.

Then tension $=\frac{W \times H}{L}$;

Tension $=\frac{20 \times 2240 \times 1}{6}=7466 \cdot 6$ lbs., due to gravity alone;

or allowing $\frac{1}{70}$ for friction,

Total tension $=\left(\frac{20 \times 2240 \times 1}{70}\right)+7466 \cdot 6=8106 \cdot 6$ lbs.

Question.—What engine power, expressed in foot-pounds per minute, is required to draw a load of eight tons up an incline of 1 in 5, at a speed of five miles per hour, excluding friction?

$$\text{Speed of rope}=\frac{5 \times 5280}{60}=440 \text{ feet per minute.}$$

$$\text{Load}=\frac{8 \times 2240 \times 1}{5}=3584 \text{ lbs.}$$

Engine power required (load in lbs. × speed in feet per minute) $=3584 \times 440=$ 1,576,960 ft. lbs. per minute.

Question.—What size of engine, length of stroke, etc., would be required to haul 400 tons per shift of nine hours, from a road 1000 yards long and dipping 1 in 12? The system of haulage to be main and tail rope, and the speed of rope six miles per hour, the diameter of the drum being 6 ft.

$$\text{Speed of rope per minute}=\frac{6 \times 5280}{60}=528 \text{ ft.}$$

$$\text{Time per trip}=\frac{1000 \times 2 \times 3}{528}=11 \text{ minutes.}$$

Time allowance for changing at each end, about 4 minutes.
Total time per trip, $11+4=15$ minutes.

Load per trip $=\frac{400 \times 15}{9 \times 60}=11$ tons (approximately).

Suppose the tubs hold 10 cwts. each, and to weigh 4 cwts. each when empty, twenty-two tubs will be required each trip, and the weight of the tubs will be $22 \times 4 = 88$ cwts. or 4·4 tons; therefore the gross load per trip will be $11 + 4{\cdot}4$, or 15·4 tons.

The load due to inclination $= \dfrac{15{\cdot}4 \times 1}{12} = 1{\cdot}28$ tons.

The circumference, C, of the rope required would be $= \sqrt{\dfrac{1{\cdot}28 \times 8}{3}} = 1{\cdot}84$ in.

Weight of rope per fathom $= C^2 \times {\cdot}9 = 3{\cdot}06$ lbs.
Total weight of rope $= 500 \times 2 \times 3{\cdot}06 = 3060$ lbs.
Taking friction of load equal to 32 lbs. per ton, and friction of rope equal to $\frac{1}{20}$th its weight; then
Friction of full load $= 15{\cdot}4 \times 32 = 492{\cdot}8$ lbs.

And friction of rope $= \dfrac{3060}{20} = 153{\cdot}0$ lbs.

Total friction is $492{\cdot}8 + 153 = 645{\cdot}8$ lbs.
Total load = full load + rope + friction = 38201·8 lbs.

Total resistance to engine $= \dfrac{(15{\cdot}4 \times 2240) + 3060}{12} + 645{\cdot}8 = 3520{\cdot}4$ lbs.

Now the work done in the engine is equal to the work done on the plane, and can be expressed by the formula:

$$D^2 \times {\cdot}7854 \times P \times L \times N \times M = \text{load in lbs.} \times \text{circumference of drum in feet.}$$

Assume the effective steam pressure to be 45 lbs. per sq. in., and the modulus M, $\frac{2}{3}$ for single engine and $\frac{4}{5}$ for coupled engines, the length of stroke being $3\frac{1}{2}$ ft.

$$\therefore D^2 \times {\cdot}7854 \times 45 \times 3{\cdot}5 \times 2 \times \tfrac{4}{5} = 3520{\cdot}4 \times 6 \times 3{\cdot}1416$$
$$\text{After cancelling } D^2 \times 15 \times 3{\cdot}5 \times {\cdot}8 = 3520{\cdot}4 \times 4$$
$$\text{or } D^2 \times 3{\cdot}5 = \frac{3520{\cdot}4}{15 \times {\cdot}2} = 1061{\cdot}3$$
$$\therefore D = \sqrt{\frac{1061{\cdot}3}{3{\cdot}5}} = 18{\cdot}3 \text{ in.},$$

the size of engines required.

This problem may also be worked out by the formula:

$$D^2 \times {\cdot}7854 \times P \times L \times N \times M = (W \times h) + (\text{friction} \times \text{cir. of drum in ft.}),$$

$$\text{or } D = \sqrt{\frac{(W \times h) + (F \times \text{cir. of drum}),}{{\cdot}7854 \times P \times L \times N \times M}}$$

where as before,

D = diameter of cylinder in in.
L = length of stroke in ft.
M = efficiency of engine.
P = effective steam pressure in lbs. per sq. in.
N = number of strokes per revolution of the drum.
W = weight of loaded train in lbs.
F = friction of loaded train in lbs. and rope.
h = vertical height which the train is raised during one revolution of the drum.

$$\text{As } h = \frac{6 \times 3{\cdot}1416 \times 1}{12} = 1{\cdot}57 \text{ ft.}$$

Taking the same figures as before, we have,

$$D=\sqrt{\frac{(15{\cdot}4\times 2240\times 1{\cdot}57)+(645{\cdot}8\times 6\times 3{\cdot}1416}{\cdot 7854\times 45\times 3{\cdot}5\times 2\times \frac{4}{5}}}=\sqrt{334{\cdot}98}=18{\cdot}3 \text{ in.}$$

Question.—Find diameter of cylinder required, length of stroke, etc., to haul 300 tons per eight hour shift from a road dipping 1 in 20, laid with tram rails. Length of road 600 fms. Tubs to weigh 4 cwts. and hold 10 cwts. Size of driving wheel 6 ft. diameter. System, endless rope with a speed of two miles per hour.

Tons per hour $=\frac{300}{8}=37{\cdot}5$: speed of rope 3520 yds. per hour.

Tubs per hour $=37{\cdot}5\times 2=75$: distance tubs will have to be apart on rope $=\frac{3520}{75}=46{\cdot}9$ or 47 yds. (for simplicity).

Number of tubs on rope at once $=\frac{600\times 2}{47}=25{\cdot}55$, approximately 26, *i.e.* 26 full and 26 empty.

Weight of loaded tubs,	$=26\times 14\times 112=40768$	lbs.
,, empty ,,	$=26\times 4\times 112=11648$	,,
Weight of rope, say at 4 lbs. per fathom	$=600\times 2\times 4=4800$	,,
	Total $=57216$	,,

Suppose the total friction is $\frac{1}{30}$th $\therefore\ \frac{57216\times 1}{30}=1907{\cdot}2$ lbs.

Load due to inclination $=\frac{\text{full load}}{\text{inclination}}=\frac{40768\times 1}{20}=2038{\cdot}4$ lbs.

Total load to be overcome by engine $=2038{\cdot}4+1907{\cdot}2=3945{\cdot}6$ lbs. And as before, work done by engine = work done on plane during one revolution of drum. Assume effective steam pressure at 50 lbs. per sq. in., and the number of strokes per revolution of drum at four.

$$\therefore\ D^2\times{\cdot}7854\times 50\times L\times 4\times \tfrac{4}{5}=3945{\cdot}6\times 6\times 3{\cdot}1416$$

$$\text{After cancelling } D^2\times L,=\frac{3946{\cdot}6\times 3}{50\times{\cdot}4}=591{\cdot}9$$

$$\text{Assume length of stroke 3 ft.}\ \therefore\ D=\sqrt{\frac{591{\cdot}9}{3}}=14{\cdot}00 \text{ in.}$$

The size of engines required would be therefore 14·00 in. diameter, with a 3 ft. stroke geared two to one.

This problem may also be worked out by the formula

$$D=\sqrt{\frac{(F\times\text{cir. of drum})+(W-w)h}{\cdot 7854\times P\times L\times N\times M}},$$

the letters having the same value as in preceding question, and in this case w = weight of empty tubs.

$$\text{Here } h=20 \text{ ft.} : 18{\cdot}84 \text{ ft.} :: 1$$

$$=\frac{18{\cdot}84\times 1}{20}={\cdot}94 \text{ ft.}$$

$$\therefore\ D=\sqrt{\frac{\{1907{\cdot}2\times 6\times 3{\cdot}1416\}+\{57216-(11648+4800)\}{\cdot}94}{\cdot 7854\times 50\times 3\times 4\times \frac{4}{5}}}=\sqrt{\frac{74272{\cdot}64}{376{\cdot}9}}=14{\cdot}00 \text{ in.}$$

CHAPTER XII.

PUMPING.

In all mines water is met with in greater or less abundance, shallow mines being, as a rule, more heavily watered than deep mines, owing to more frequent occurrence and greater width of the cracks or fissures in the overlying strata by which water can reach the workings, and to the absence therefrom of impermeable beds. In deep mines, which are seldom troubled with much water, such firm impermeable beds of strata are always present to some extent in the overlying strata, and prevent the water from entering the workings.

The different methods of draining mines of water are by means of adit levels, tanks or chests, siphons and pumps.

Adit levels can only be utilised under certain conditions, such as when the mine is situated on the side of a hill, or where the workings are at a higher level than some parts of the surrounding country. These conditions rarely exist in connection with coal-mining, unless in the opening out of virgin coal-fields in hilly countries. In metalliferous mining, adits are, however, much used, many of them being of great length and costing large sums of money. In the Freiberg district of Saxony there is an adit level 8½ miles long, or, including branches, 25 miles in length, which cost about £360,000, the length of time occupied in this important work having been thirty-three years.* The Halkyn adit in Flintshire is about four miles in length, and drains a large area to a depth of 230 yds. It is estimated that this adit is now discharging 15 million gallons per twenty-four hours, or a total weight of 66,000 tons, the whole of this water being a natural flow.

Tanks or Chests.—If the quantity of water is not great and the pit is deep, it would be expensive to lay down a special pumping plant, while the water may be raised by water tanks or chests, especially if the winding engines are not fully employed drawing coal. The quantity of water dealt with in this way should not exceed 30 to 40 gallons per minute throughout the twenty-four

* *Ore and Stone Mining*, Sir C. Le Neve Foster, sixth edition, p. 462.

hours, and this should be regarded as the maximum which should not be exceeded.

Where there is a considerable quantity of water to be dealt with, pumping will be found the cheapest method. In drawing water in tanks great injury is often done to the winding ropes, through part of the water falling back into the shaft and washing the grease or lubricating oil off them, and also by the great strain they have to undergo in lifting heavy tanks full of water. The load is often very much heavier than the usual load of coal. The dipping of the tank into the water causes 'slack,' while the vibration of the rope causes repeated bending to occur just above the capping, which tends to injure the rope, and if not carefully watched may result in breakage.

Siphons.—The siphon is not applicable in the same way that a pump is, since by the former, water must always be delivered at a *lower* level than that of the receiving end of the pipes. We may, therefore, define a siphon as being ***an apparatus for conveying a liquid from a higher to a lower level over an intervening height.***

In construction the siphon is a simple piece of apparatus, and consists of a U-shaped pipe, one limb of which is longer than the other. The short limb dips into the liquid to be siphoned, and the other discharges it at a lower level. A vacuum being continually formed by the escape of water from the longer limb, the pressure of the atmosphere, acting on the free surface of the water into which the shorter limb dips, forces it up the latter, when, having reached the highest point of the column, it descends by gravity with a velocity proportionate to the difference of level between the outlet and the free surface of the source of supply.

Since the action of the siphon depends on the atmospheric pressure it is obvious the height to which the water can rise will never be greater than that of the water barometer *at the time*, which at greatest is about $33\frac{3}{4}$ ft., no matter what the amount of fall may be at the discharge end. In practice the height to which the water will rise will not be more than 26 or 27 feet, the difference being due to the friction of the water in the pipes, but it will be better if the vertical height does not exceed 20 or 22 feet.

To start the flow of water in the siphon, the two ends must be closed by plugs or taps. Water is then poured in at the highest point until the pipes are filled; this opening is then closed and the receiving and discharge ends are opened. The water in the pipe discharging produces a vacuum, thereby setting up a continuous flow. A better and more economical arrangement is to place a small hand-pump on the siphon at the highest point of the pipes, to pump the air out, and thus allow the water to rise.

The air and gases held in water are liberated on moderate reduction in pressure with great ease; and as nearly all water contains more or less dissolved gas, this will be liberated in the siphon at its highest point, and may accumulate there until the pressure equals

that due to the acceleration head, when the siphon will cease to flow.

In laying down a siphon the greatest care should be exercised, so that it will have an opportunity of working under the most favourable conditions. The pipes should be laid with a regular gradient all the way to the highest point, and the pipes should be of sufficient diameter for the water to flow with a velocity that may reduce the friction to a minimum. The joints should be thoroughly air-tight; if the siphon has to continue working for any length of time, the joints ought to be run with lead, as this will be the most satisfactory and cheapest way in the end. A 'tail clack' should be put on at the bottom of the receiving end to prevent the pipes from getting empty when the siphon stops running.

Pumps.—The best and most usual method for raising water from mines is by pumping. The first point of importance is the capacity of the plant required. In deciding this it is necessary to ascertain, as nearly as possible, the maximum quantity of water likely to be met with both in the shaft and in the workings. In sinking a shaft in a new and untried district it is impossible to do so, but in districts which have been well opened out it can often be done without much difficulty. The plant should be capable of raising a larger quantity of water than any ascertained maximum, so that a sudden inflow could be dealt with, if necessary, to prevent the flooding of the workings.

Pumps for raising water in mines are generally of the reciprocating type and may be classified as: (1) Plunger or ram pumps; (2) piston pumps; and (3) bucket or lift pumps. Other kinds of pumps are also employed, such as the centrifugal and Fontigaine pumps, but these can only be adopted for limited lifts. The power employed is either (*a*) steam, (*b*) compressed air, (*c*) water or hydraulic pressure, (*d*) electricity.

Conditions affecting the Working of Pumps.—The working of pumps is influenced by various conditions, such as: (1) The height at which they are placed above sea level, *i.e.* atmospheric pressure conditions. (2) The temperature of the water. (3) The size and length of delivery and suction pipes. (4) The area, weight, and lift of valves.

These conditions have all to be carefully considered in designing or deciding upon the necessary pumping plant for a given position either on the surface or underground. The chief requirements mining pumps are expected to fulfil are: (*a*) They should be capable of working for long periods with little repair, packing, or adjustment. (*b*) They should be capable of being operated under water (a particularly desirable feature in sinking pumps). (*c*) They should be capable of passing sandy or dirty water, and sometimes acid water, without too rapid deterioration or corrosion. (*d*) Their speed and capacity should be easily adjustable to suit the varying inflow of water.

Pump Fittings. Pipes.—The pipes used in connection with pumps may be made of wood, cast iron, or wrought iron or steel.

Wood pipes are seldom if ever used in Britain for pumping, but in some parts of the United States, and where the climate is extremely variable, they are much used for certain descriptions of work, when the pressures are light. For pressures not exceeding 85 to 90 lbs. per sq. in., or a head of water equal to about 200 ft., they are said to be economical. The advantages claimed for them are that they contract and expand to only a small extent, and are therefore well suited for climatic changes, while they offer little resistance to the flow of water, and do not decay readily if water is kept constantly flowing through them. They are built up with staves, much like ordinary barrels, and are strongly bound with wrought-iron hoops.

Cast-iron pipes were formerly almost exclusively used for pumping, and are so still to a very large extent, but for certain classes of work steel or wrought-iron pipes are displacing them. The great disadvantage of cast-iron pipes is, that where they have to be of a large section, they are very heavy and difficult to handle. The joints in this class of pipe are usually made with a common flat flange, and an india-rubber ring inserted between them, the joint being well secured by nuts and bolts. For deep lifts and heavy pressures the top flange should have a groove cut in it, while on the bottom flange a rib should be cast to fit into this groove. An india-rubber joint of circular section is fitted into the groove and the rib on the other pipe fitted on it, the two ends being well screwed down with nuts and bolts in the usual way.

Wrought-Iron and Steel Pipes.—Pipes of this class are now extensively used for pumping purposes, and possess considerable advantages over those made of cast iron. They are more easily handled, and also cheaper, while sections of any reasonable length can be readily cut off and fitted wherever required. It has been found, however, in practice, that when dirty or acid water has to be dealt with, that cast-iron pipes are best. Pipes of large section should be made of mild steel, which is more homogeneous and possesses greater strength than wrought iron. The joint used for this class of pipe is somewhat different from that used in cast-iron pipes. Leaded joints are sometimes used when the pipes are permanently fixed, such as in siphons, but they are not suitable for shaft work.

Eadie's joint, which is somewhat similar, is also much used both for water and compressed air. At Kladno, Bohemia, a flange packing has been successfully adopted for a head of water of 1700 ft. One of the flanges is recessed to admit a ring of rubber or metal of L shape, this ring being held in position by a rigid metal ring, which gives great security and tightness to the joint (fig. 349).

Expansion Joints.—Where the column of steel or wrought iron used is very long in either shafts or inclines, expansion joints should

be used to prevent 'elbowing.' The commonest forms are merely joints with an ordinary stuffing-box, containing hydraulic packing. The pipe entering the stuffing-box should be perfectly smooth, and kept well greased.

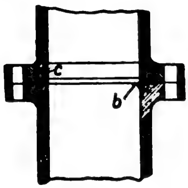
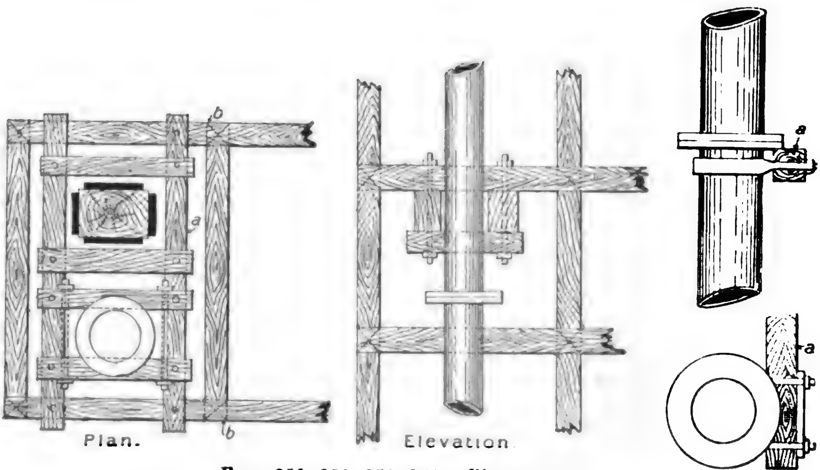


Fig. 349.

Pipe Supports.—Pipes in a vertical shaft ought to be properly supported to keep them in a vertical line. It is also necessary to contrive some support for the weight of a long column of pipes. A support ought to be put in every eight or ten fathoms at least. A common method is to lay timber pieces across the shaft, and bolt them firmly down to the buntons at one end, and fix them into the strata at the other.

Sometimes the rods are supported by the method shown in figs. 350, 351. Two cross pieces ***a a*** are laid across the pieces *b b*, the two sets of timbers being firmly bolted together with hanging and horizontal bolts. When there is little room in the shaft, the method



Figs. 350, 351, 352, 353.—Pipe supports.

employed is to use a gland, bent to the circle of the pipes, with two screwed ends, over which a plate of iron is fixed. A piece of wood *a*, like a saddle, may be fixed between the pipe and the support, and the gland then tightened by the nuts (figs. 352, 353).

Size of Pipes.—The diameter of pipes will, of course, depend on whether they are connected to a lift or force pump, and on the velocity at which the water is required to flow. For ordinary lift pumps the pipes ought to be ½ in. larger than the working barrel, so that the bucket can be drawn through them if necessary, and changed at the top of the lift or on the surface. With force or

plunger pumps this is unnecessary, and therefore, for this class of pump, the diameter of the delivery pipes may be smaller.

The velocity of the flow of water in pipes varies as the area and area $\propto D^2$, therefore velocity $\propto D^2$.

$\therefore d_1^2 : d_2^2 :: \surd_1 : \surd_2$ or $\frac{d_1^2}{d_2^2} = \frac{\surd_1}{\surd_2}$; where d_1 = diameter of plunger.

d_2 = diameter of rising main.
$\surd_1$ = velocity of water in rising main.
$\surd_2$ = velocity of plunger.

If the maximum velocity of the water in the rising main is not to exceed say 150 ft. per minute, and the speed of plunger 60 ft. per minute, diameter of plunger 15 in., what should be the diameter of the rising main?

Here $d = 15$ in., $\surd_1 = 150$ ft. per min., $\surd_2 = 60$ ft. per min. $\therefore \frac{15^2}{d_2^2} = \frac{150}{60}$

$\therefore 150 d_2^2 = 60 \times 15^2 \therefore d_2 = \sqrt{\frac{60 \times 225}{150}} = \sqrt{90} = 9{\cdot}48$ in. So that pipes about 9½ in. diameter would be suitable for a 15 in. diameter plunger under the above conditions. A practical rule is to make the rising main about ⅔ the area of that of the plunger.

Thickness of Pipes.—The thickness of pipe required will vary according to the pressure to be sustained and the constancy or otherwise of the pressure. If the pressure is uniform, the pipes may be made much lighter than where it varies, by the stopping and starting of the water column, as in a single-acting pump. Similarly, if the water is corrosive the pipes will require to be extra strong.

Pipes are liable to rupture either transversely or longitudinally. Their strength may be calculated according to the following formula:

Rupture longitudinally: $P \times D = 2t \times f$.
Rupture transversely: $P \times D^2 \times {\cdot}7854 = t \times D \times 3{\cdot}1416 \times f$.
Where P = pressure in lbs. per sq. in. t = thickness of pipes in in.
D = internal diam. of pipes in in. f = coefficient of metal employed (cast iron = 16,~~000~~, wrought iron = 50,000, steel = 70,000).
500

EXAMPLE.

Given a set of pipes 10 in. diameter (internal) and ⅝ in. thick: find the pressure required to rupture them (*a*) transversely, and (*b*) longitudinally.

(*a*) *Longitudinally:*

$$P \times D = 2t \times f$$
$$P \times 10 = 2 \times \tfrac{5}{8} \times 16500$$
$$\therefore P = \frac{2 \times 5 \times 16500}{10 \times 8} = 2000 \text{ lbs. per sq. in.}$$

(*b*) *Transversely* :

$$P \times D^2 \times \cdot 7854 = t \times D \times 3 \cdot 1416 \times f$$
$$P \times 10 = 4 \times \tfrac{5}{8} \times 16500$$
$$\therefore P = \frac{4 \times \tfrac{5}{8} \times 16500}{10} = 4000 \text{ lbs. per sq. in.}$$

For safety, the working pressure would be taken at $\frac{1}{6}$th to $\frac{1}{8}$th of the above pressures.

The thickness of pipes required can be obtained from the formula deduced from above, $t = \dfrac{PD}{2f}$.

Example.

Given a head of water of 600 ft., and internal diameter of pipes 10 in. ; what thickness would the pipes require to be, taking this safe working pressure at $\frac{1}{8}$th ?

$$t = \frac{600 \times 434 \times 10}{2 \times 18500 \times \frac{1}{8}} = \cdot 56 \text{ in. or } \tfrac{9}{16}\text{th in.}$$

Where the pipes are exposed to shock as in mine pumps, the practice is to make them with a greater thickness of metal than would be brought out by the foregoing formulæ, which are based on the assumption that the pressure exerted is constant, and which takes no account of defects in casting, etc. The following rule will be found to coincide more closely with practice :—

$$t = (\cdot 00022\ Pd) + \cdot 15\sqrt{d} \text{ (Molesworth).}$$

Where t = thickness of metal in inches or decimals of an inch.
P = pressure in lbs. per sq. in. due to head of water $= h \times \cdot 434$.
d = dia. of pipe in inches.

Taking the example as already worked out :—

$$t = (\cdot 00022 \times 600 \times \cdot 434 \times 10) + \cdot 15\sqrt{10} = \cdot 5728 + \cdot 4740 = 1 \cdot 046 \text{ in.}$$

For working barrels $t = (\cdot 00017\ Pd) + 1 \cdot 26$.

To find the necessary thickness for wrought-iron pipes with a given head of water :—

$$t = \cdot 0000288\ hd$$
$$\text{or } t = \cdot 0000665\ Pd$$

Weight of Cast-iron Pipes.—$W = 2 \cdot 45\ (D^2 - d^2)$

Where W = weight in lbs. per ft. of pipe.
D = external diam. of pipe in inches.
d = internal " "
$2 \cdot 45$ = constant.

The weight of two flanges may be taken as approximately equal to one foot of pipe.

In practice the thickness of pipes used is somewhat greater than the calculated thickness, to provide for wear and other contingencies. The following thicknesses of metal are used for cast-iron pipes of different diameter and for different heads of water :—

Inside Diameter of Pipe.	50 ft. Head or 21·65 lbs. Press.	100 ft. Head or 43·3 lbs. Press.	150 ft. Head or 64·85 lbs. Press.	200 ft. Head or 86·6 lbs. Press.	250 ft. Head or 108·25 lbs. Press.	300 ft. Head or 129·9 lbs. Press.
	Thickness of Metal.	Thickness of Metal.	Thickness of Metal	Thickness of Metal.	Thickness of Metal.	Thickness of Metal.
ins.	ins.	ins.	ins.	ins.	ins.	ins.
8	0·422	0·450	0·474	0·498	0·522	0·546
10	0·459	0·489	0·519	0·549	0·579	0·609
12	0·491	0·527	0·563	0·599	0·635	0·671
14	0·524	0·566	0·608	0·650	0·692	0·734
16	0·580	0·604	0·625	0·700	0·748	0·796
18	0·589	0·643	0·697	0·751	0·805	0·859
20	0·622	0·682	0·742	0·802	0·862	0·922
24	0·687	0·759	0·831	0·903	0·975	1 047

Pump Valves or Clacks.—Valves for pumps used in mines are of various types, their design and construction depending upon whether the water is clean or gritty, acid or otherwise corrosive, and whether the temperature is high or low.

There are three types of valves generally used, viz.:—

Hinged valves, commonly called clacks.
Straight lift valves, which rise vertically on their seats.
Flexible valves, which alter their form on opening.

In ordinary pumps the valves mostly used are of the hinged type, with either a single or double lid (see figs. 354, 356, 358).

For direct-driven pumps, straight lift valves are very largely used. Figs. 356, 357 show such a valve, which is commonly used for any pressure up to 500 lbs. per square inch. Double flap or 'butterfly' lid valves are almost entirely used for buckets and clack valves of lifting sets. Thus the latter valve is very suitable for lifts which do not exceed 30 to 40 fms.; but for lifts over 30 fms., iron hinges should be used instead of leather. Figs. 359, 360 show this type of valve.

The valves are usually faced with leather, but where acid water has to be pumped, rubber is to be preferred, and sometimes vulcanite or brass is used instead of iron for mounting. If the water is hot a rubber composition is used for facing them.

Hinged valves are more liable to leakage than straight lift valves, as they wear more unequally; but the tightness of a valve depends a good deal on the pressure per square inch on its face, while the less bearing it has, the more difficult it is to keep it tight. On the other hand, the larger the bearing surface, the greater the wear. Leakage with hinged valves will probably amount to 10 or 12 per cent., and for lift valves 5 to 7½ per cent.

A valve should not be larger than 10 in. in diameter. If larger

outlets than this are required they should be made double, for when the valve is very large it is impossible to prevent it from 'hammering,' owing to the weight of water above. Good valves should be simple in construction, and not liable to get out of order. They

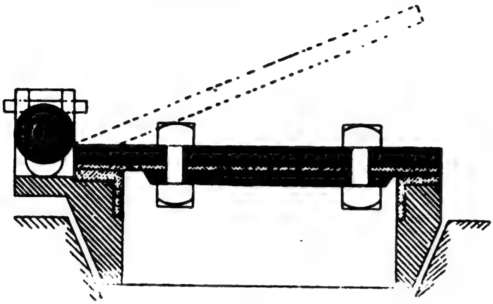


FIG. 354.—Elevation.

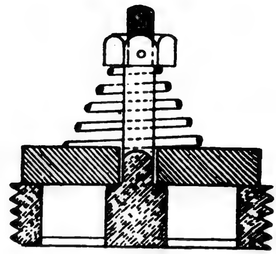


FIG. 356.—Elevation.

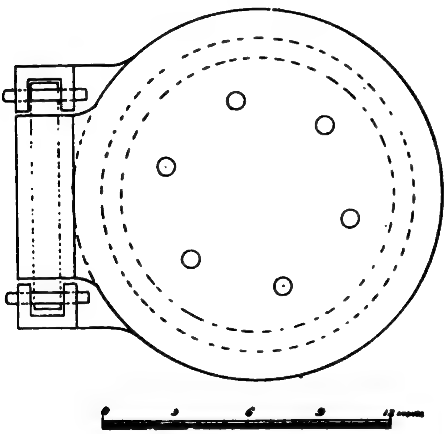


FIG. 355.—Plan.
Common Hinged Clack Valve.

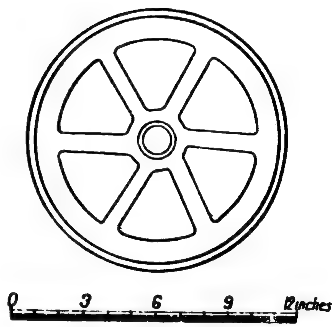


FIG. 357.—Plan.
Straight Lift Valve.

FIGS. 354, 355, 356, 357.—Various types of valves.

should present as little resistance as possible to the flow of water, and should close as quickly as possible on the completion of the stroke.

Buckets.—Buckets are usually constructed of iron with leather mountings, the shell being often composed of gutta-percha, or a composition containing this. They are provided at the top with two flap lids opening from the centre. Their construction will be under-

stood from figs. 361, 362, 363. The leather and other mountings should be of the very best quality, otherwise they will rapidly wear, and cause much trouble and annoyance. For very high lifts the bucket is sometimes made entirely of metal. Fig. 364 shows such a bucket constructed of steel, and simply made to fit sufficiently well to prevent leakage, and not occasion too much friction.

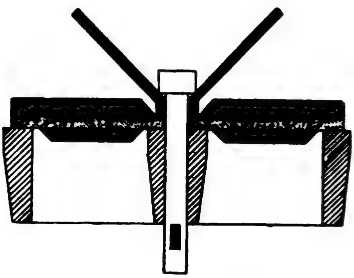


FIG. 358.—Double hinged valve.

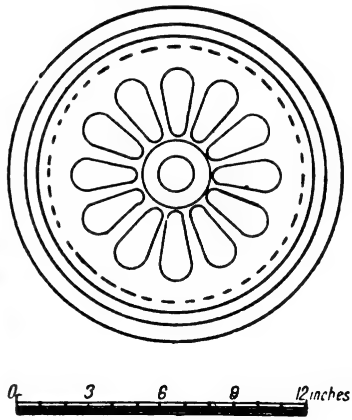


FIG. 360.—Flexible valve plan.

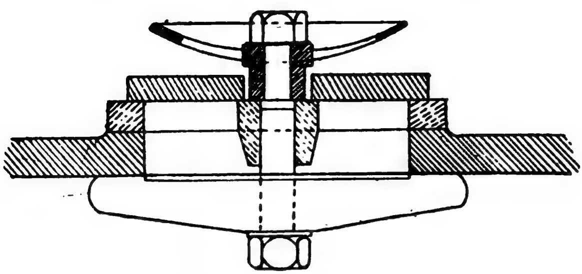


FIG. 359.—Flexible valve elevation.

FIGS. 358, 359, 360.—Various types of valves.

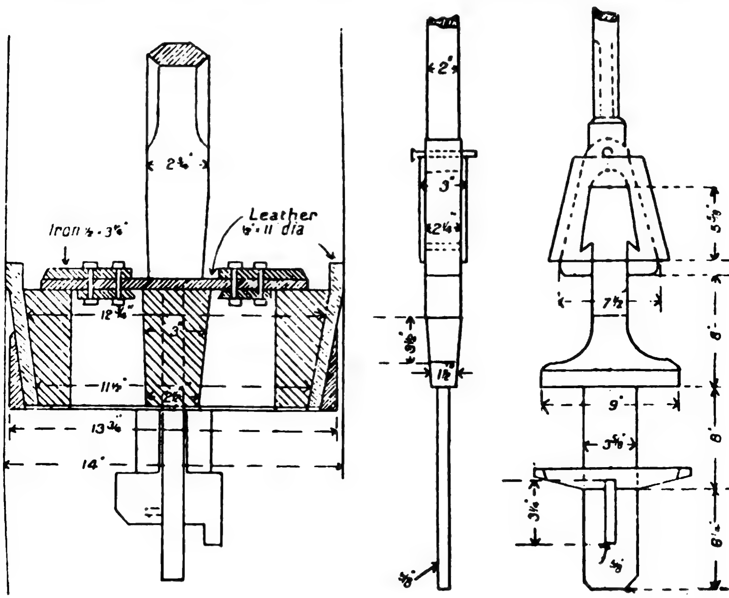
The bodies of buckets of this class are made a good deal longer than ordinary sorts, which increases their efficiency against leakage. Grooves are sometimes turned in the body of the bucket, and brass rings inserted to make them work smoothly.

Plungers or Rams.—Plungers for ordinary pit pumps are usually made of cast iron; for small pumps, or where the water contains much acid, they are sometimes made of brass or lined with it. The plunger is usually made hollow, and connected to the rods by the

methods shown in figs. 365, 366. In large hollow plungers the wood rod is sometimes simply driven in firmly to give it sufficient gripping power to prevent it from slipping. The plunger should be accurately turned and finished smooth, so as to work with as little friction as possible.

Plungers should be kept well greased at the top of the stuffing-box, as this will reduce friction and tend to easy working.

Pump-Rods or Spears.—Pump-rods are usually made of pieces of pitch pine or oak, 18 to 45 ft. long, 30 ft. being a common length, and square or rectangular in section, joined together by strong iron



Figs. 361, 362, 363.—Bucket and connections.

plates to withstand the heavy strains to which they are subjected. Iron rods are sometimes used, but not very extensively, at least in Britain. They are more difficult to make and to put together, get more easily out of order, and are more difficult to repair than wooden rods. Speaking of iron pump-rods, Callon says: "The lightness of rods is not usually a point to be desired, because we are often led, on the contrary, to give them additional weight. We may fairly conclude that this is one of those cases more frequent in practice than we think, in which the word *modification* need not necessarily signify *improvement*, and that metallic rods have no decided superiority over wooden ones."

On the whole, rods made of pitch pine have been found to be in every way better suited for pumping. The size of section will vary according to the size of pump and length of lift. The pieces are joined one to another either by a common square joint or a scarf joint, and held together by iron plates and bolts (see figs. 367, 368,

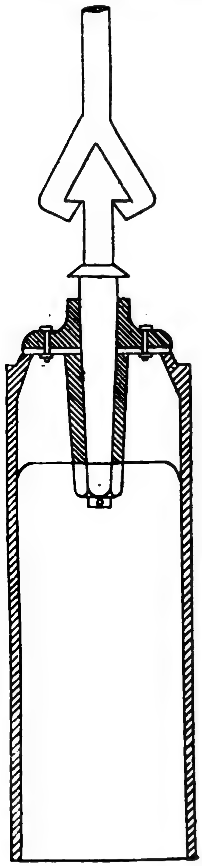
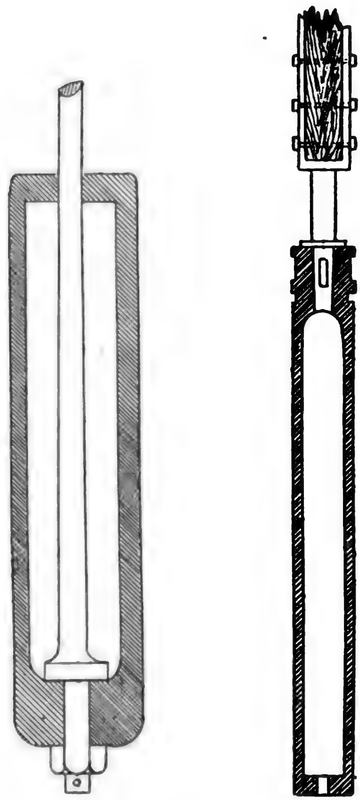


FIG. 364.—Steel bucket.

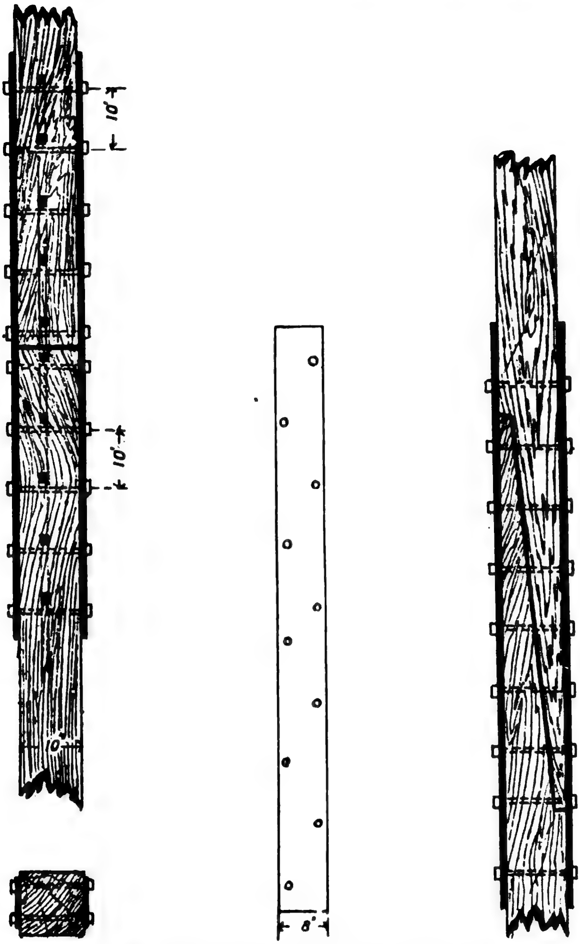


FIGS. 365, 366.—Plunger connections.

369). For plunger or force pumps the joint should be made square, as the whole of the pressure or work done is practically on the down stroke; but for bucket or lift pumps, where the tension is greatest on the up stroke, the joint is often scarfed in a zigzag fashion which is found to give better results.

For rods up to 8 in. or 9 in. square, two straps or plates and

'dumb' or 'clink' bolts, put in at right angles to the plate bolts and a little above them (fig. 367), will be sufficient to prevent the other bolts from tearing along the grain of the wood. The bolt holes in the plates should be bored in zigzag fashion, otherwise if they are



FIGS. 367, 368, 369.—Pump-rods or spear joints.

placed in a line they will have a greater tendency to split the rod. For rods over 9 in. square four plates will be necessary to give the requisite strength. The plates used on the joints are from 6 to 18 ft. in length, according to the size of the rods and the length of lift;

they may be either uniform in section or somewhat tapered at each end. Additional strength may be obtained by overlapping the ends of the rods, under the jointing plates, by a separate piece fitted in, and holding the ends by means of steel keys (*a*, fig. 370).

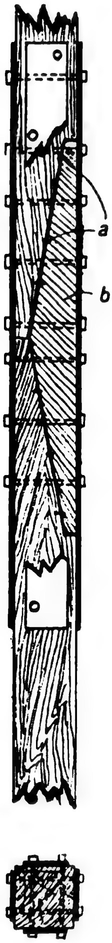
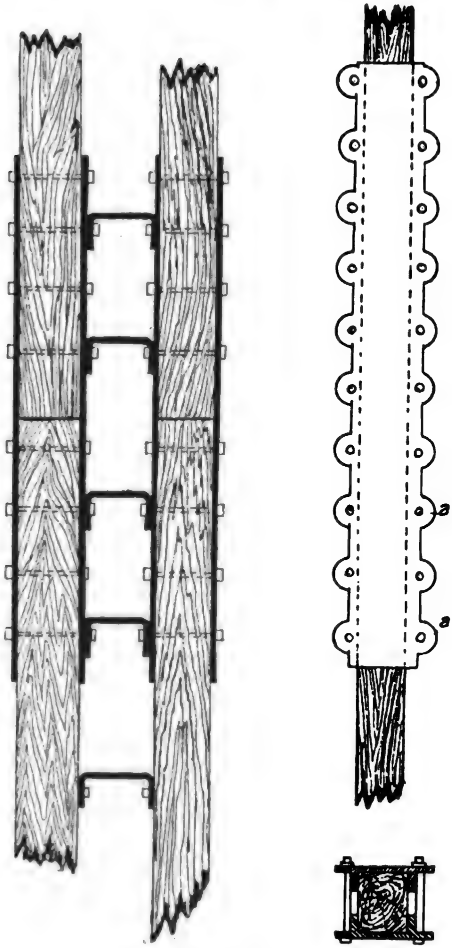


FIG. 370.



FIGS. 371, 372.

When the rods required for a pump have to be very large in section, and would be too large to be handled easily, two rods of smaller section are sometimes employed, braced together by iron stays and bolts, as in fig. 371.

Sometimes the lengths of pump-rods are joined together by plates having projections for the bolts (fig. 372), to obviate the necessity of boring bolt holes in the rods themselves. This plan is said to give additional strength to the rods, and saves labour in boring holes, and the plates are more easily put on. The same object may be attained by using two long wooden straps at the joint and bolting them firmly on the rod.

In both these methods great care should be taken that none of the bolts get slack, as the rods would be liable to drop out from between the plates.

The sizes of rods used, as already mentioned, will depend on the size and speed of pump, and length of lift or head of water. The following sizes are often used in practice, viz. :—

8-in. bucket. Lift 30 to 40 fms. Rods 4 in. square, plates ⅜ in. × 3½ in. × 8 ft. Rod bolts ¾ in. diameter. 'Dumb' bolts ½ in. diameter. Bolts put in 10 in. centre to centre.

10-in. bucket. Rods 5 in. square, with same sizes of plates and bolts as for 8-in. bucket.

12-in. bucket. Rods 6 in. square, plates ½ in. × ¼ in. × 9 ft. Rod bolts ⅞ in. diameter, dumb bolts ⅝ in. diameter, and five on each side of joint.

16-in. bucket. Rods 8 in. square, plates ½ in. or ⅝ in. × 6 in. × 9 ft. Rod bolts 1 in. diameter, dumb bolts ¾ in. diameter.

With a 20-in. bucket the rods would require to be 10 in. square, and four plates would have to be used instead of two, the plates being about ¾ in. × 8 in. × 12 ft., and the bolts 1¼ in. to 1½ in. diameter. Pitch pine sawn into lengths for pump-rods costs about 2s. 6d. per cubic foot. Plates bored and ready for putting on cost 8s. per cwt., and bolts 10s. 6d. per cwt.

In calculating the size of rods and plates to be used for any given size of pump and length of lift, different formulæ are used, giving somewhat different results, but a simple plan is to equate the opposing strains or pressures to which the rods are subjected.

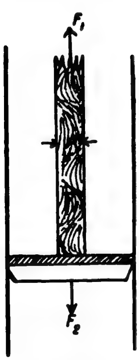


FIG. 373.

In an ordinary lifting-pump as illustrated (fig. 373), let f_1 be the tensile strength of pitch pine (12,000 lbs. per sq. in.) and A the sectional area of rod in square inches. Then the total *upward* strain on rod will equal $A \times f$.

If a is the area of bucket in sq. in. ($D^2 \times \cdot 7854$), and p the pressure in lbs. per sq. in., due to head of water (head in ft. × ·434), then $a \times p$ will be the total *downward* strain.

Now $A \times f$ should equal $a \times p$ or $F_1 = F_2$.

This rule would give the data required for calculating the size of rod necessary to exactly balance the downward strain, but in actual practice a large margin would have to be allowed for safety, this factor depending on the speed and size of pump. The inertia of the water column and of valves would also have to be taken into account.

"In large forcing pumps, with the engine placed on the surface, all the excess of the weight of the rods above that of the column of water, plus the friction, should be balanced, save a little surplus which is left to make the downward stroke begin distinctly. Not infrequently the main line of rods is over-weighted

on purpose, but at the same time balanced, in order to increase the mass set in motion. The object of this is to produce a moderate acceleration only at the commencement of the upward stroke when the steam has its full pressure and the power is much greater than the load, for it is on the rate of this acceleration that the reactions of the force of inertia which are got up in the various parts depend, and these reactions again increase the strain on the entire system." *

It is generally allowed that the useful weight of the main rod ought to exceed by $\frac{1}{10}$th to $\frac{1}{8}$th the weight necessary for raising the delivery valves in plunger pumps. For all practical purposes we may take the power necessary to overcome the friction of the water in moving through the pipes and valves, and other contingencies, at about $\frac{1}{4}$th of the power required to set the water column in motion, *i.e.* the total pressure exerted on the ram. To cover this the factor of safety should be comparatively high, say 30 to 40 for quick-running, intermittent pumps, where the strokes are more frequent and the strain greater than in large slow-moving pumps, where the factor need not exceed 20 to 30.

Example.—What size of (*a*) rod and (*b*) plates should be used in a 20-in. lifting set, the head of water being 40 fms., and the factor of safety 30?

(*a*) $A \times f = a \times p \times M$.

$A \times 12000 = 20^2 \times \cdot 7854 \times 240 \times \cdot 434 \times 30 = 81 \cdot 8$ sq. in. (area of rod required), or about 9 inches square.

(*b*) The size of iron plates may be found in the same way, by substituting 50,000 † for 12,000, the factor of safety being taken at 10. If two plates are put on each joint and each plate is 6 in. broad,

$$A \times 50{,}000 = 20^2 \times \cdot 7854 \times 240 \times \cdot 434 \times 10\,;$$

$$5\,A = 32 \cdot 64. \quad \therefore A = 6 \cdot 52 \text{ sq. in. area; hence}$$

$$\text{Thickness of plate} = \frac{6 \cdot 52}{2 \times 6} = 0 \cdot 54 \text{ in. (about } \tfrac{5}{8} \text{ in.).}$$

The plates would therefore require to be $\frac{5}{8}$ in. × 6 in. × 12 ft. with bolts $1\frac{1}{4}$ in. diameter. For plunger pumps the compressive strain (6000 lbs. per sq. in.) of pitch pine would require to be taken instead of tensile strain.

The following empirical formulæ are also used for determining the dimensions of pump-rods, plates, and bolts, where D = diameter of pump in inches.

Bucket Pump-Rods:

$$\text{Sectional area of rods} = \frac{\text{area of bucket}}{3\frac{1}{2}} \quad . \quad . \quad . \quad . \quad . \quad . \quad (a).$$

$$\text{,, \quad ,, \quad plates} = \frac{\text{area of bucket}}{D+7} \quad . \quad . \quad . \quad . \quad . \quad . \quad (b).$$

$$\text{,, \quad ,, \quad bolts} = \frac{\text{area of rod}}{10} \quad . \quad . \quad . \quad . \quad . \quad . \quad (c).$$

Plunger Pump-Rods:

$$\text{Sectional area of rods of lowest lift} = \frac{\text{area of pump}}{1\frac{1}{2}} \quad . \quad . \quad . \quad . \quad . \quad . \quad (d).$$

$$\text{,, \quad ,, \quad second ,,} = \frac{\text{area of pump}}{2} + \text{area of bottom lift} \quad . \quad (e).$$

$$\text{,, \quad ,, \quad third ,,} = \frac{\text{area of pump}}{3} + \text{area of second lift} \quad . \quad (f).$$

$$\text{,, \quad ,, \quad plates} = \frac{\text{area of rods}}{10} \quad . \quad . \quad . \quad . \quad . \quad . \quad (g).$$

$$\text{,, \quad ,, \quad bolts} = \frac{\text{area of rods}}{10} \quad . \quad . \quad . \quad . \quad . \quad . \quad (h).$$

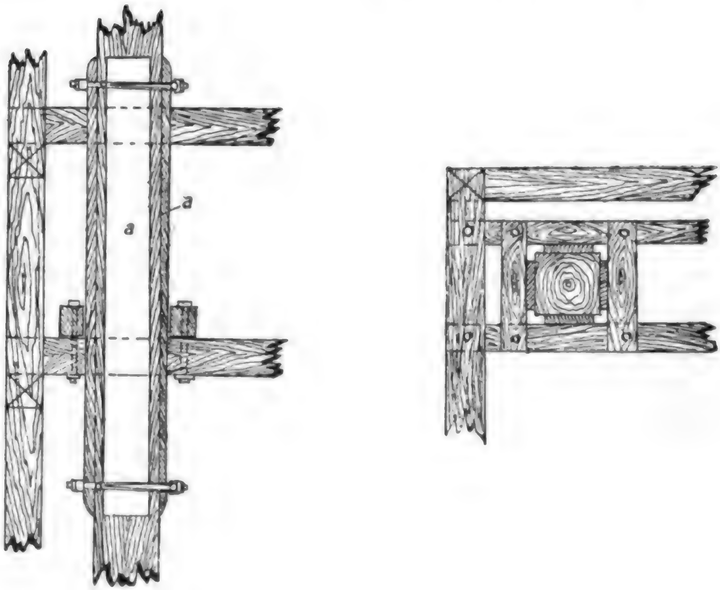
These rules are based upon the assumption that the lift is not greater than 60 to 80 yards.

---

* *Lectures on Mining*, by M. Callon, vol. ii. p. 363.

† Tensile strength of wrought iron = 50,000 lbs. per sq. in.

Guiding the Pump-rods.—When the pump-rods are in operation, they require to be kept from twisting or vibrating in the shaft; this is done by 'collaring' or guides. The collaring is composed of pieces of wood nailed or bolted to the cross buntons at certain fixed distances apart. They ought to be placed sufficiently near each other to keep the rods from 'buckling' under compressive strains, and they ought to be kept in line or as nearly so as possible. Pieces of hard wood, oak, or beech, called cleats, are fixed to the rods (*a a*, fig. 374) either by nails, counter-sunk bolts, or by glands at the top and bottom.



Elevation. Plan.

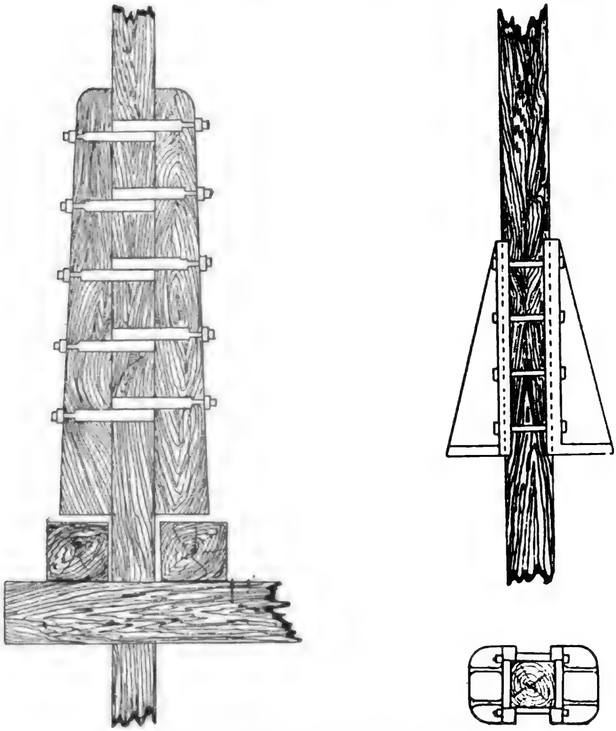
FIGS. 374, 375.—'Collaring' for rods.

These cleats are 2 in. to 3 in. thick, tapered at the ends, and made a little longer than the stroke of the pump. They are put on to prevent the rods from wearing too rapidly, and to prevent breakage. Figs. 374, 375 show the methods of collaring.

Sometimes a piece of wood, square in section, is fixed to the rods and works in a 'shoe' fixed on the cross buntons or the guides, or the rods may be made of iron and a 'shoe' fitted to suit, which is much more substantial.

All these cleats or rubbing pieces should be kept well lubricated, to reduce friction and wear to a minimum. The rods working inside the pipes (called 'wet' rods) must also be provided with cleats to prevent the bolts or plates from rubbing against and wearing the pipes.

'Bang' Pieces.—The rods should also be provided with 'bang' pieces, by means of which, in the event of the rods breaking or getting detached from the engine for repairs, they may be caught and rested on strong beams placed across the pit (called 'horse trees'). The 'bang' pieces (figs. 376, 377, 378) may be made of either wood or iron clamped to the rods. It is best to secure these catches to the rods without the bolts passing through the latter, because then



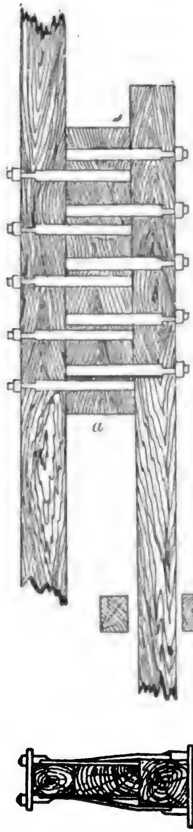
FIGS. 376, 377, 378.—'Bang' pieces.

they can slip a little, and so lessen the shock if they happen to fall away. With the same object a 'cushion' is sometimes used, made of thin boards (*a a*) placed immediately above the 'horse trees.' The cross beams or 'horse trees' require to be sufficiently strong to withstand the shock of the column of rods falling on them suddenly.

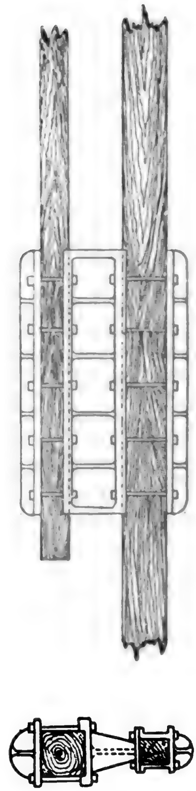
There ought to be at least two of these supports, one immediately below the 'bell cranks,' and the other at the bottom of the rods.

Offsets or Aprons.—The usual method of operating two sets of

pumps by one column of rods is generally done by means of offsets called aprons. The usual way is to insert a square block of wood, called an 'apron,' between the two sets of rods, so as to carry the one set (those of the lower lift) clear of the working barrel of the other set (figs. 379, 380). Cast-iron aprons are also used for making



Figs. 379, 380.—Offset or 'apron.'



Figs. 381, 382.—Cast-iron 'apron.'

offsets (figs. 381, 382), and are better than wood, as they are less liable to shrink, and the clamps less likely to get loose. They are, however, more expensive and more difficult to fix.

Guides or collaring should be placed as near where the offset is made as possible, both above and below.

The offset may also be effected by using a 'cross-head,' which is to be preferred to the ordinary apron, as the set of rods operated

below the cross-head is in direct line with the rods working above, so tending to reduce friction and strains (see fig. 383).

Balancing the Rods.—In most pumping arrangements where wooden rods are used some balancing arrangements are required, as the rods are nearly always heavier than the column of water they have to raise.

Callon says: "This excess of weight requires the expenditure of a certain amount of motive power to raise it, and as soon as the rod in its descent has opened the clacks it becomes useless and even

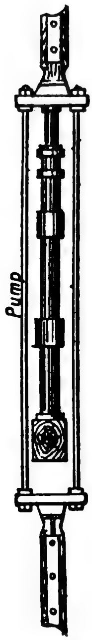


FIG. 383.—Cross-head.

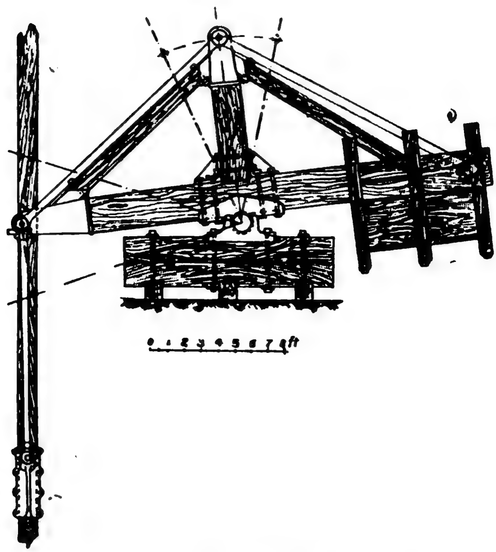


FIG. 384.—Balance beam.

injurious. In fact, it becomes necessary to counterbalance it, so that the rod may not acquire an accelerated velocity." *

There are different methods of applying a counterbalance, but the commonest one is to use a balance on the 'bell-crank' at the surface, or an auxiliary bell-crank in some part of the shaft.

A double bell-crank is sometimes used for the counterbalance, and at one end a heavy weight is placed, generally a large box filled with scrap metal, which can be adjusted whenever required (fig. 384).

Hydraulic or loaded pistons are also used for balancing the rods,

* *Lectures on Mining*, M. Callon, vol. ii. pp. 347–8.

and compressed air, where used, can also be applied for this purpose, and is to be preferred to the hydraulic piston.

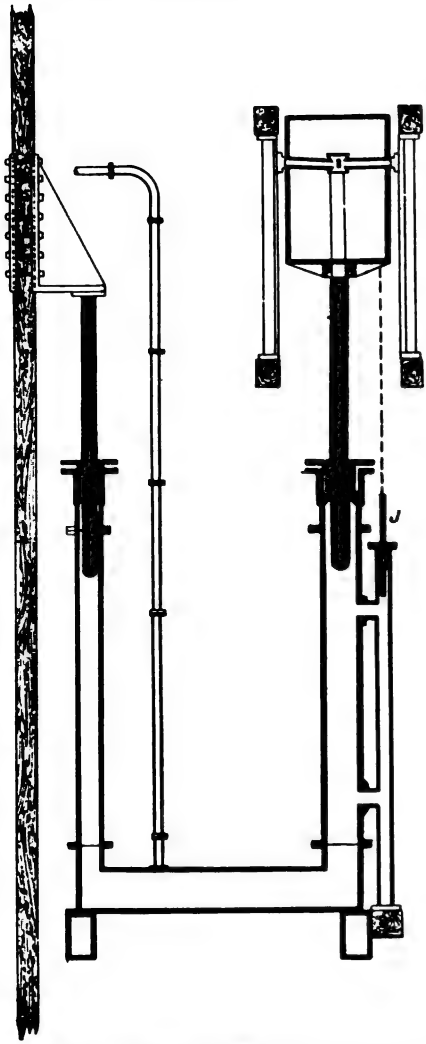


FIG. 385.—West and Darlington's balance.

In West and Darlington's hydraulic balance an auxiliary piston is used, worked off the main rods (fig. 385). The loss of water is made up by the pipe *h*, which communicates with a cistern placed in the shaft. When this cannot be done a small plunger J draws up and forces in the necessary supply. The same arrangement can be adopted with compressed air.*

Bucket Pumps. — These pumps are largely used for raising water in shafts. They are similar to ordinary well pumps. A working barrel *a* (fig. 387), smoothly bored out, works the bucket *b*. Immediately below the working barrel is the clack piece *c*, which contains the suction or receiving clack *d*, and dipping into the water is the wind-bore or snore-pipe *e*, which is a pipe closed at one end with a number of holes bored in it for the admission of the water.

Action of Pump.—Suppose the bucket to be at the end of the down stroke, and to be now raised. This upward motion tends to produce a vacuum below it, and the air contained in the wind-bore opens the suction valve and rushes into the working barrel. The elastic force of this air being thus diminished, the atmospheric pressure

* For fuller description of this balancing arrangement, see *Ore and Stone Mining*, Sir C. Le Neve Foster, sixth edition, p. 476.

will cause the water to rise in the pipes until the pressure of the column, increased by that of the air inside, exactly counterbalances the pressure of the external atmosphere. At the next stroke of the bucket a fresh quantity of air will escape, and the water will rise higher in the pipes, this process being repeated until the water reaches the bottom of the bucket, provided the distance is not greater than 34 ft. In practice this height is, however, never attained, 20 to 25 ft. being the practical working distance between the bucket and the surface of the water. The pipes being now filled with water up to the bottom of the bucket, and the pump-rod falling at the down stroke, the suction valve will be closed and the valves on the top of the bucket will be open, allowing the water, which on the up stroke will be carried the full length of the stroke, to rush through. This process is repeated until the water reaches the surface. It will be seen that nearly the whole weight of water is raised on the up stroke.

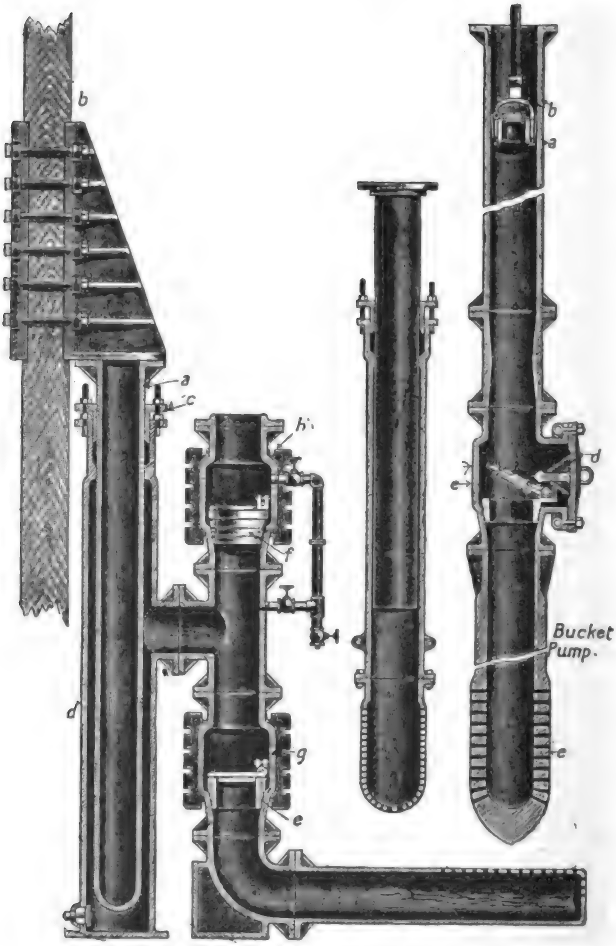
Plunger Pump.—The construction of the plunger pump is somewhat different to that of the bucket pump. In this pump there is a long solid piston or plunger *a* (fig. 386) connected to the pump-rods *b*, working through a stuffing-box *c*, in the barrel or ram casing *d*. Instead of a single valve, as in the bucket pump, there are two, a suction valve *e* and a delivery valve *f*, situated in the two clack chambers *g* and *h*.

Action of Plunger Pump.—The principle of this pump is exactly like that in the bucket pump, but it is worked somewhat differently. On the up-stroke of the plunger the water forces the suction clack open and fills the space between it and the bottom of the plunger; on the down stroke of the plunger the suction valve is closed and the water is forced up through the delivery clack the full length of the stroke, the same process being repeated at each stroke until the water is delivered at the surface. Thus the whole of the work done is practically carried out in the down stroke.

In a bucket pump the work is more unequally divided than in a plunger lift; but with a bucket pump the rods last much longer unless the water is acid and destroys the iron connections at the joints. The latter has also the advantage that in case of a sudden inflow of water in the shaft, the bucket may be drawn through the pipes and changed at the top of the lift, if, as is usual in fitting up bucket lifts, the pipes are a little larger than the working barrel. It is, therefore, generally the rule where there are a series of lifts in the shaft, to make the bottom one a bucket lift. It is usual to arrange bucket lifts with two bell-cranks, so that when one of the lifts is making an up stroke the other is making a down stroke. In this way they are more or less balanced, and the work done by the engine is better distributed.

Compared with a bucket lift the advantages of plunger pumps may be thus stated:—(1) The rods or spears being working outside

the pipes, can be more easily examined and got at for repairs when needed. (2) Any leakage at the plunger is more easily seen and repaired. (3) The delivery pipes may be smaller. It is usual to determine the size of the delivery pipes by the velocity at which the



FIGS. 386, 387.—Plunger and bucket pumps.

water can travel without excessive friction. (4) The power is more evenly distributed during the up and down strokes.

Length of Lifts.—The length of lift depends somewhat on the point at which the water is coming into the shaft, and the suitable

ness of the strata for a seat for the column of pipes and also for a lodgment for the water. For bucket pumps it is found that the most suitable length of lift is from 30 to 35 fms., and for plunger pumps 40 to 80 or 90 fms., according to the type and size of engine used to operate the pumps. With engines of the Cornish type, the lift is not often more than 45 fms., while with direct-acting engines of Bull type the lift may be 70 or 75 fms. With such long lifts the speed must be slow and the stroke long, so as to avoid too severe shocks.

With the power engine placed at the bottom of the shaft the water may be forced up any height of lift between 100 and 1500 ft.; but with very long lifts the joints at clack chambers and between the pipes will be very difficult to keep, and may give much trouble.

Speed of Pumps.—This will vary much, according to the size and type of pump used. For either plunger or bucket pumps working with rods, a good average speed is 60 to 80 ft. per minute, which may be increased a little with pumps below 12 in. in diameter. For pumps working direct without rods, the speed may be 100 to 150 ft. per minute, according to the size of the engine. In all pumps great care ought to be taken to avoid sudden shocks.

Length of Stroke.—This will also be determined by the size and type of pump used, and may vary from 3½ ft. for 8 in. or 10 in. diameter pumps, and small engines up to 12 or 13 ft. for 20 in. to 30 in. diameter pumps and large direct-acting engines.

Quantity of Water delivered by a Pump.—The quantity of water delivered by a pump can be calculated from the formula,

$$G = \frac{D^2 \times \cdot 7854 \times L \times N \times 6 \cdot 25}{144} \qquad \text{or } G = D^2 \times \cdot 034 \ L \times N \left\{ \frac{\cdot 7854 \times 6 \cdot 25}{144} = \cdot 034 \right\}$$

Where D = diameter of pump in inches.
L = length of stroke in feet.
N = number or effective strokes per minute.
G = Gallons delivered per minute.
6·25 = number of gallons in 1 cub. ft. of water.

These formulæ will give the theoretical quantity delivered, 5 per cent. to 12 per cent. of which should be deducted for slip at the valves and bucket or plunger.

Example.—How many gallons could a pump 15 in. diameter, with a 6 ft. stroke, and making eight effective strokes per minute, deliver per minute, allowing 10 per cent. for slip?

$$\begin{aligned} G &= D^2 \times \cdot 034 \times L \times N \\ &= 15_2 \times \cdot 034 \times 6 \times 8 \\ &= 369 \cdot 2 \text{ gallons per minute without slip.} \end{aligned}$$

$$\therefore \ G = \frac{367 \cdot 2 \times 90}{100} = 330 \cdot 48 \text{ gallons per minute allowing for 10 per cent. slip.}$$

The theoretical quantity delivered per foot stroke is approximately found by the expression $G = \frac{30}{D^2}$.

The diameter of pump required for a certain quantity of water may also be derived from the above formula,

$$D^2 = \frac{G}{\cdot 034 \times L \times N}.$$

Example.—What size of pump would be required to deal with 10,000 gallons of water per hour? Engine to work 124 hours per week, and 10 per cent. to be allowed for leakage.

10,000 gallons per hour = 166·6 gallons per minute.
124 hours per week = 17·7 hours per day.

$\therefore \frac{166 \cdot 6 \times 24}{17 \cdot 7} + 10$ per cent. $= 248 \cdot 48$ gallons per minute. If the effective speed of pump is 30 ft. per minute, then $D^2 = \frac{248 \cdot 48}{\cdot 034 \times 30} = 15 \cdot 6$ in. diameter. The size of pump required would therefore be 15·6 in. diameter, say with 5 ft. stroke, going at six strokes per minute.

Plant Required for Working Pumps.—The engines used for operating pumps may be worked with steam, compressed air, hydraulic pressure, or electricity.

For operating pumps in shafts, engines worked by steam are almost universally used, especially when they are situated on the surface. Steam engines for working pumps by means of rods may be divided into three classes: rotative, non-rotative, geared.

The non-rotative types may be either direct-acting or indirect-acting. Direct-acting engines are those in which the piston-rod is in line and connected to the pump-rods. The Bull engine is the most familiar type of this class of pumping engine.

Sometimes a direct-acting engine is defined as a "machine for raising water, so constructed that the pump is worked by the motive-power cylinder, without the intervention of beams, connecting rods, cranks, or fly-wheels."

Direct-acting engines are not so largely used now as formerly, unless for large quantities of water, or deep shafts, because the cylinder usually obstructs the mouth of the shaft. This is often inconvenient, unless there is plenty of room, or a shaft is wholly set apart for pumping. This type of engine also requires very heavy and expensive foundations, and it is also wasteful of power, owing to the uncertainty of the point at which the stroke is completed, and the necessity for thus working with a large 'clearance' space in the cylinder.

Geared pumping-engines are very largely used for actuating pumps with rods, and are receiving more attention from engineers than formerly. The great advantage of this class of engine is that the speed can be easily altered to suit any conditions and that the work per stroke can be varied much more easily than in other types.

It must not be forgotten, too, that steam, being a much more elastic body than water, can be moved more rapidly without danger, and hence in a geared engine the steam may be moving very rapidly,

while the water may be moving very slowly. This class of engine can likewise be erected more cheaply and disposed of more readily than large direct-acting engines, if not required for, or readily adapted to, other work than pumping.

Special Types of Pumping Engine:—*Bull Engine.*—The Bull engine may be termed a Cornish engine, with a beam working under the cylinder instead of over it (fig. 388). The cylinder is placed directly over the shaft in a line with the pump-rods, these being a continuation of the piston-rod, and acting direct.

The great objection to this class of engine is, that it blocks up the shaft and requires large and expensive foundations. On the other hand, the first cost is less than for an ordinary Cornish engine.

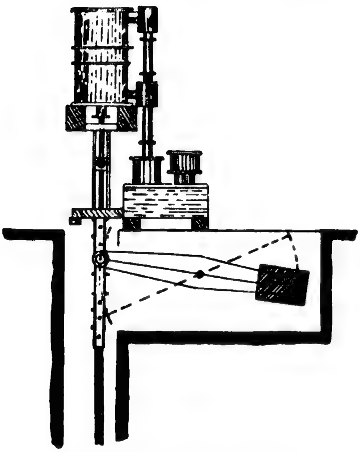


FIG. 388.—Bull engine.

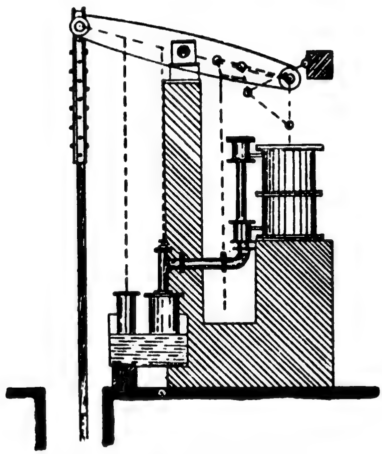


FIG. 389.—Cornish engine.

A modification of the Bull engine is sometimes used, in which, to avoid the cylinder being placed directly over the shaft, it is placed some distance back and bell-cranks introduced between the piston and the pump-rods. In this way the space in the shaft is not encroached upon to the same extent, and less expensive foundations are necessary.

Sometimes these engines are worked on the compound condensing principle, with high- and low-pressure cylinders. The steam is regulated in the same way as in the Davey engine, by an auxiliary piston and tappet valves, and is not permitted to act by expansion alone, but by *wire-drawing* to a considerable extent.

Cornish Engine.—This is one of the oldest of pumping engines, the type having been first devised by Watt, and such engines are still used to a considerable extent for pumping in mines (fig. 389). The Cornish engine is generally worked as a single acting, high-pressure

condensing engine. It consists of a vertical cylinder, having the piston-rod connected to one end of a large 'walking' beam, and the pump-rods to the other end. The beam usually works on a fulcrum near its centre, thus causing the stroke of the engine to be somewhat longer than that of the pumps, and the flow of water in the pipes is therefore less than the speed of the steam piston, in order to lessen the shock of the water in the pipes on the sudden closing of the valves at the reversal of the stroke. In the working of the engine steam is admitted at the top of the cylinder, this causing the piston to make its down stroke. The steam is then cut off and an equilibrium valve opened by means of a tappet-rod, which allows the steam to pass to the under side of the piston, thus equalising the pressure and causing a pause in the movement. The weight of the rods alone is sufficient to cause the down stroke and force the water up. The steam is then exhausted into the condenser, where a vacuum is formed by the air-pump. This process is repeated at each stroke. The engines are best suited for slow speeds and long strokes. They are very expensive at first, but once set up they give little trouble, require little repair, and give a high efficiency.

Barclay Engine.—This is another well-known type of the beam pumping engine, which is much used for dealing with large quantities of water. In construction it is a modification of the Cornish engine. It consists of a single steam cylinder placed vertically, the piston rod being connected to the overhead beam. To one end of the beam the pump-rods are connected, while the other end works on a fulcrum which takes the form of a rocking pillar. Near to the centre of the beam a rod connection is attached for working the air pump and condenser hot-well. This type of engine, like the Cornish, is best suited for large quantities of water and where a long stroke can be adopted. Owing to the arrangement of the beam the stroke of the engine is less than that of the pumps: for instance, the piston stroke may be 8 ft., while the pump stroke may be 10 ft. Engines of this type are made with cylinders 50 to 100 in. diameter and 8 to 12 ft. stroke, moving at two to four strokes per minute.

Another type of engine much used for pumping in the Fifeshire coal-field, where large quantities of water are met with, is one in which a pair of cylinders, high- and low-pressure, working compound and condensing, are placed vertically directly over the shaft, the piston rods of the cylinders being connected direct to the ends of the bell-cranks which operate the pump rods. These engines are well suited for raising large quantities of water, 1000 to 1500 gallons per minute, from depths of 150 to 250 fms. At the present time a plant of this type, with cylinders 58 and 100 in. respectively, is being put down to raise 1200 gallons from an expected depth of 350 fms. The objections to this type of pumping engine, like the Bull engine, are that the cylinders are placed directly over the shaft, and they take up a great deal of space, necessitating the sinking of

a very large shaft, or devoting a shaft entirely to pumping. They are best suited for large rectangular shafts where the necessary space can be set apart for them at one end of the shaft.

Wherever large pumping engines of the types just described are employed it is almost imperative that a duplicate plant be installed to deal with the water in the event of any serious breakdown. This duplicate plant may consist of a direct-acting pumping engine, such as the Riedler pump, placed underground at the bottom of the shaft to force the water direct to the surface.

Horizontal Pumping Engines.—A typical pumping plant of this description is at work at Boghead Colliery, Bathgate, N.B. The plant is designed to deal with 500 gallons per minute from a depth of 160 fms. It consists of high- and low-pressure cylinders placed tandem (fig. 390) 38 in. and 76 in. diameter respectively, each

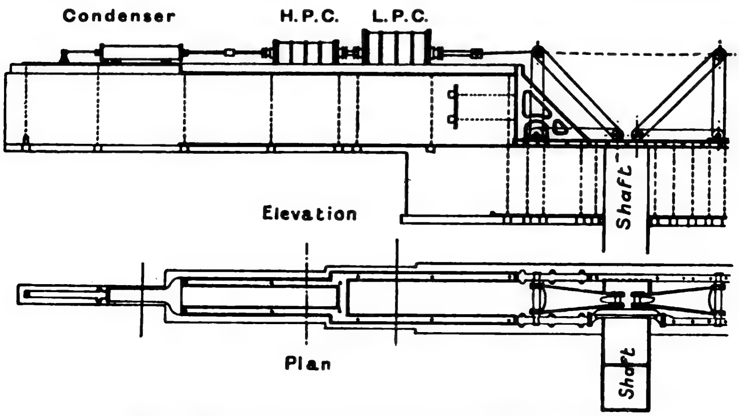


FIG. 390.—Direct-acting horizontal pumping engine.

cylinder having a stroke of 9 ft. The piston-rod of the low-pressure cylinder is continued forward and connected to the bell-cranks, while the piston-rod of the high-pressure cylinder is continued backwards to the condenser and air-pump. In this case no fly-wheel is used. The pumps are worked by a double set of rods attached to the bell-cranks at the surface, each set of rods working two plunger and one bucket lift at three different stages in the shaft. At the top and mid lifts the offsets are made by means of cross-heads bolted to the rods and connected to each other by iron side rods far enough apart to clear the pumps. The details of the pumps are as follows :—

Two bucket lifts at pit-bottom, 160 fms. from surface, each set $14\frac{1}{2}$ in. diameter, lifting 30 fms. ; two plunger lifts at 130 fms. each $19\frac{1}{2}$ in. diameter, lifting 56 fms. ; two plunger lifts, each 20 in.

diameter, at 74 fms., lifting to the surface. The sizes of pump-rods for the various lifts are:—bottom bucket lifts, rods 6 in. square; second lift, rods 12 in. square; top lift, rods 14 in. square; the iron strapping plates at the joints for these two latter are 11 × ¾ in. × 17 ft. The available steam pressure is 90 to 100 lbs. per square inch, plus 12½ lbs. vacuum from condenser.

A good type of pumping engine is one in which high- and low-pressure cylinders are used. The connecting rods to the bell-cranks are in a direct line and form a continuation of the piston-rod. A back piston-rod works a fly-wheel which imparts steady motion to the engine. This class of engine has much to recommend it for pumping purposes, as it works very smoothly, and can be easily regulated to any required speed.

Geared pumping-engines are very largely used for pumping water at collieries, and, as has already been pointed out, are peculiarly fitted for such work, and have many advantages to recommend them. They may be constructed either with one, two, or four cylinders, and may be condensing and compound. The gearing can be altered to suit any speed and any size of pump. This easy adaptability to work under varying conditions makes these engines very suitable for mining, and gives them many advantages over the larger types of vertical engines. An engine constructed in this way gives a high efficiency and a small consumption of steam, and works very smoothly at almost any reasonable speed. Fig. 391 shows the arrangement of a geared engine with low- and high-pressure cylinders, as used at the Priory Colliery, Blantyre, where the water has to be raised from a depth of over 200 fms. The high-pressure cylinder is 20 in. diameter, and the low-pressure cylinder 34 in. diameter, with 5 ft. stroke geared to about 2½ to 1. The cylinders are steam-jacketed, and work with a very small consumption of steam. They can be regulated to any speed, as low as 1½ strokes being attainable.

At Holm Colliery, Kilmarnock, the pumping engine is constructed with quadruple cylinders, two high-pressure and two low-pressure ones being supplied. The high-pressure cylinders are 16⅞ in. and 15⅞ in. diameter respectively, while the low-pressure cylinders are 24¼ in. and 26 in. diameter respectively, the stroke being 3½ ft. and the gearing 5½ to 1. The water is raised the first 68 fms. by two rams 15 in. diameter and 5 ft. stroke, the lower lift of 13 fms. being two buckets 10 in. diameter and 5 ft. stroke. The speed is five strokes per minute, and the water raised 420 gallons per minute.

Direct-acting Steam Pumps Underground.—Direct-acting pumping engines placed underground are very largely superseding surface engines with heavy pump-rods working in the shaft. The greater simplicity of construction and cheapness combined, make them desirable for pumping purposes, and although, as a rule, they require

much fuel, this is compensated for in other ways. These engines are nearly always double-acting, and are directly coupled to the pump. They may also be made to work geared, but this arrangement is not so much employed underground as the direct-acting.

Double-acting pumps have generally a comparatively short stroke, with very much smaller masses to move them than pumps actuated

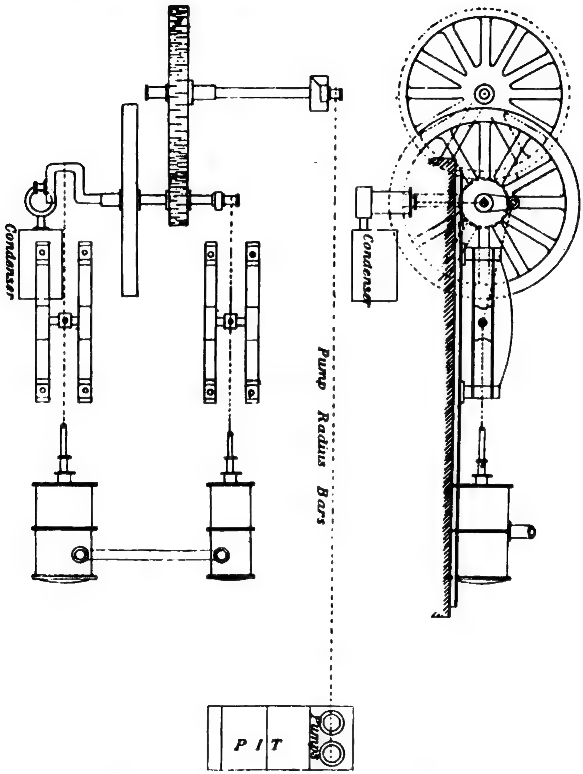


FIG. 391.—Geared pumping engine.

from the surface with rods, and they can therefore be driven at much greater speeds.

The higher speeds at which they can be driven, combined with their smaller size, are the main features which commend this class of pumps for use underground. Direct-driven steam pumps are now used for heads of water varying from 200 to 1800 ft.

Where engines are placed underground, a large lodgment, sufficient for, at least, a week's water, ought to be provided, to give time for repairs on the engine should they be necessary. With engines placed

underground, the following are some of the main advantages to be gained :—

The necessity for heavy pump-rods, which are expensive at first cost and for upkeep, is done away with.

As no pump-rods are required, less shaft space is occupied.

The engines can be worked at high speeds, as there are no heavy rods to put in motion at the commencement of each stroke, and the work is evenly distributed.

The flow of water being continuous, smaller pipes can be used in the shaft.

On the other hand, there are the following disadvantages to be kept in view :—

Steam must be taken underground or boilers fitted up in the workings, both of which methods are more or less objectionable; especially the latter.

The engine is liable to be drowned by a sudden inflow of water (the same, however, may be said of any other type of pump operated from the surface).

Loss of steam by condensation and difficulty of dealing with the exhaust steam.

Increase in temperature of air if placed near the intake, and consequently injurious effects from the moisture to the roof and timber of roadways.

The use of steam in confined places is attended with danger.

Piston Pumps.—This type is largely used for direct-acting pumps working underground without rods, and is invariably double-acting.

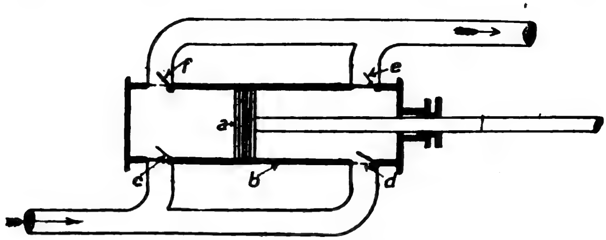


FIG. 392.—Piston pump.

The working of the pump will be understood from fig. 392. A piston *a* works in the barrel *b*, to which are four openings fitted with valves, two receiving and two discharge. The action of the pump is as follows :—When the piston moves outward, the clacks or valves *c* and *e* will, supposing the pump is full of water, be open, water being received through the valve *c* and discharged through the valve *e*. On the return or inward stroke the valve *d* will be open, receiving water, and the valve *f* open discharging, so that there is a continual flow of water during both strokes.

There are so many steam pumps in the market, all equally suitable for underground work, that it is difficult to select any particular

make for detailed description. The general principle of a double-acting steam pump will be seen from fig. 393, in which the steam cylinder is connected direct to, and is in line with, the double-acting piston pump, provided with an air vessel on the delivery column.

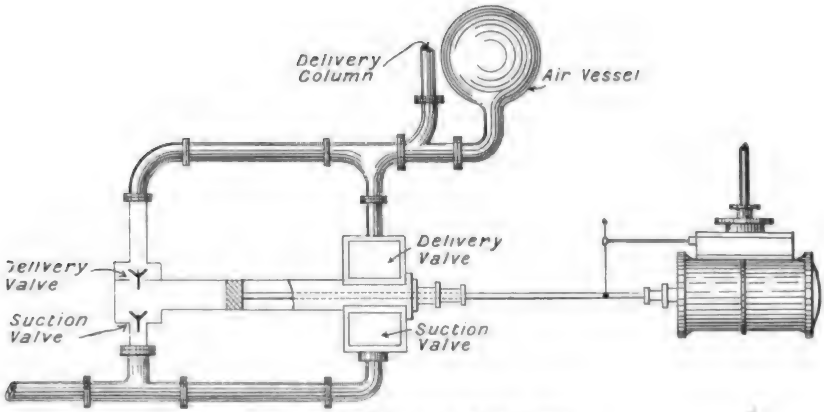


FIG. 393.—Double-acting steam pump.

Worthington Pump.—This is a well-known type of direct-acting steam pump for underground work. It consists of a steam cylinder with piston (A, fig. 394), the piston-rod of which is continued and

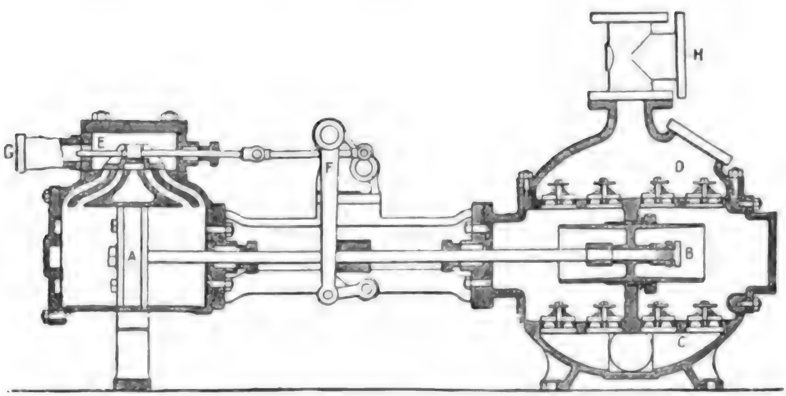


FIG. 394.—Worthington steam pump.

directly connected to a double-acting plunger B, in the water cylinder or pump. The steam is admitted to the cylinder by an ordinary slide valve E, working upon a flat face over the port-holes. The valve receives its motion from a vibrating arm F, which swings through

the whole length of the stroke, with long and easy leverage. As the moving parts are always in contact, the blow inseparable from the tappet system is avoided. The double-acting plunger B works through a deep metallic packing ring, bored to an accurate fit. Both the ring and the plunger can be quickly taken out, and either refitted or exchanged for new ones; and if it is desired at any time to change the proportions between the steam pistons and pumps, a plunger of smaller or larger diameter can be readily substituted. The water enters the pump from the suction chamber C, through the suction valves, then passes partly around and partly by the end of the plunger, through the force valves, nearly in a straight course, into the delivery chamber D. The valves usually consist of small discs of rubber or other suitable material. The plunger is located some inches above the suction valves, to form a settling chamber, into which any foreign substances may fall below the wearing surfaces. Two steam cylinders and two pumps are usually cast together to form one machine. The right-hand division moves the steam valve of the left hand one, and *vice versâ*; under this arrangement one pump takes up the motion when the other is about to lay it down, thus keeping up a uniform delivery of water.

Riedler's Pump.—The main feature in this pump is that the valves are aided in their movements by mechanical means, and so nearly perfect action is secured. The valves are constructed so as to open freely without any mechanical aid, but a little before the time when they should close entirely, when the velocity of the water is considerably reduced so that a partial closing will offer no appreciable obstruction, a lever or rod operated by valve gear from the crank-shaft moves forward and closes the valve; the arm then recedes and removes the pressure from the valves before the time for opening arrives. The valve gear is constructed in various ways, and may be operated by levers, cams, or eccentrics. Fig. 395 shows the main outlines of the pump. These pumps are driven direct, and although they cannot be driven at such high speeds as some other types of steam pumps, they can be worked very successfully for very high lifts up to 2000 ft., and pump 600 to 800 gallons of water per minute. A Riedler pump has recently been installed as a duplicate to the large pumping engine on the surface at the Aitken Pit, Kelty, Fife, to deal with 1500 gallons per minute against a head of over 1200 ft.

Davey's Differential Engine.—This engine is also largely used for pumping purposes. The main requirements to be satisfied in a good pumping-engine are economy in consumption of fuel, safety in working, and immunity from stoppages.

The distribution of steam should be effected in such a way as to cause no shock or slip in the pumps; and in the event of a sudden easing of the load, the engine should be safe; in short, it should be self-governing under extreme variations of conditions. To secure this, the Davey Differential Valve Gear, which admits steam to the

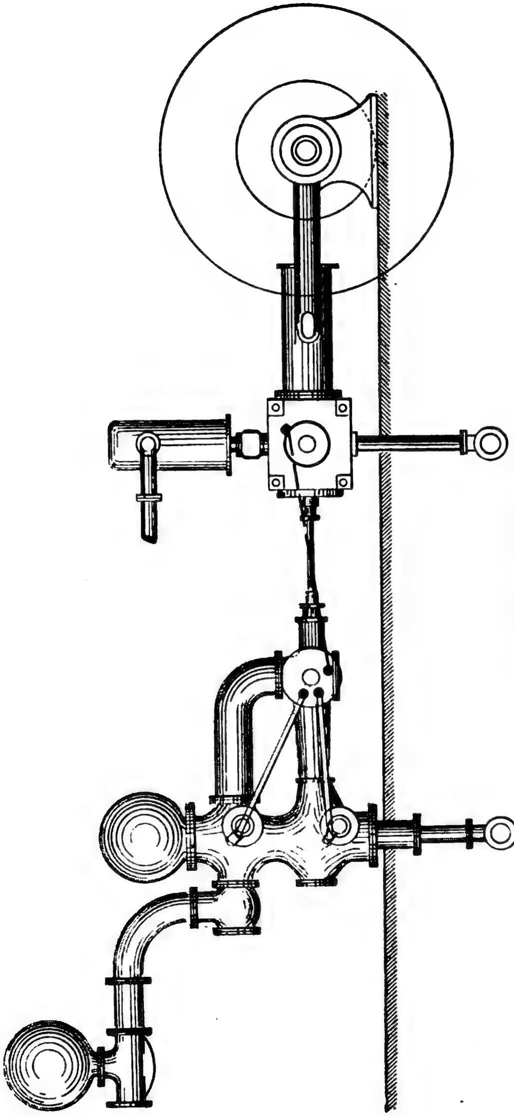


Fig. 395.—Riedler's pump.

engine in proportion to the resistance to be overcome, and, in case of a sudden total loss of load, reverses the steam to catch the piston, was designed.

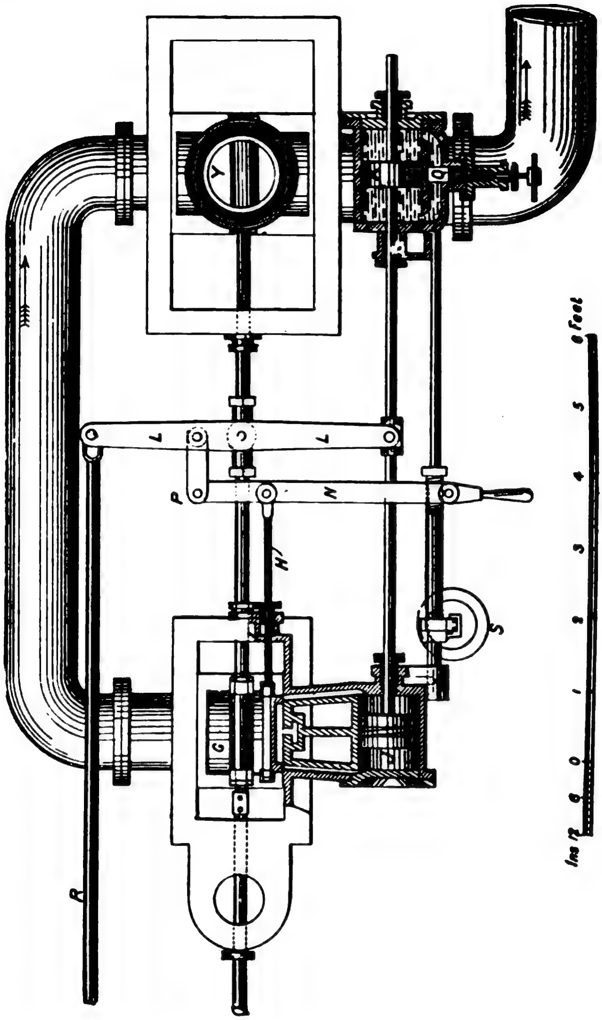


FIG. 396.—Davey's differential engine.

The action of the gear will be understood from fig. 396.

The main slide valve G is actuated by the piston-rod through a lever H working on a fixed centre, which reduces the motion to the

required extent and reverses its direction. The valve spindle is not coupled direct to this lever H, but to an intermediate one, L, which is jointed to the first lever at one end; and the other, M, is jointed to the piston-rod of a small subsidiary steam cylinder J, which has a motion independent of the engine cylinder; the slide valve I being actuated by a third lever N, coupled at one end to the intermediate lever L, and moving at a fixed centre P at the other end.

The motion of the piston in the subsidiary cylinder J is controlled by a cataract cylinder K on the same piston-rod, by which the motion of this piston is made uniform throughout the stroke, and the regulating plug Q can be adjusted to give any desired time for the stroke. The intermediate lever L has not any fixed centre of motion, its outer end M being jointed to the piston-rod of the subsidiary cylinder J; the main valve *a* consequently receives a differential motion compounded of the separate motions given to the two ends of the lever *c*. Thus the cut-off can be suited to different loads which may be on the engine.

Riedler Differential Pump.—A differential pump is practically a double-acting pump with only two valves. By an arrangement of the parts, an equal amount of work can be done on each side of the steam piston during one revolution. In a double-acting pump four valves are required, viz., one suction and one discharge valve for each end of the double-acting plunger. A differential pump has the advantage of always being primed, as will be seen by referring to fig. 397, where the column pipes D, the discharge space C, and the differential plunger chamber are always in connection. Thus the total pressure due to the water column is always on the differential plunger H. In other words, as long as there is water in the pipe, the pumping engine will always have resistance to overcome, even should suction be deficient. The arrangement, therefore, prevents undue severe hydraulic stresses on the different parts.

As it has only half the number of valves, this form of pump is simpler than the double-acting pump, and is used, in all cases, until the capacity becomes too great, so that the valves are cumbersome. When this condition obtains, it is better to use a double-acting pump instead, which would be half the size of a differential pump of the same capacity.

In working, the water enters the suction pipe A, passes into the suction air-chamber, and thence into the suction funnel B. When the main plunger J moves towards the steam cylinder, it draws in its displacement of water through the suction valve E, and on its return stroke, the suction valve having been mechanically closed, it forces a volume of water equal to its own bulk through the discharge valve F, half of this water passing out into the main pipe D, the other half passing down and following the differential plunger H. The discharge valve F being now closed, the main and differential plungers, which are connected by means of side rods, again move

towards the engine, the main plunger drawing the water through the suction valve as before (E), the differential plunger thus displacing a body of water equal in bulk to its displacement, and forcing it through the discharge pipe C into the main D.

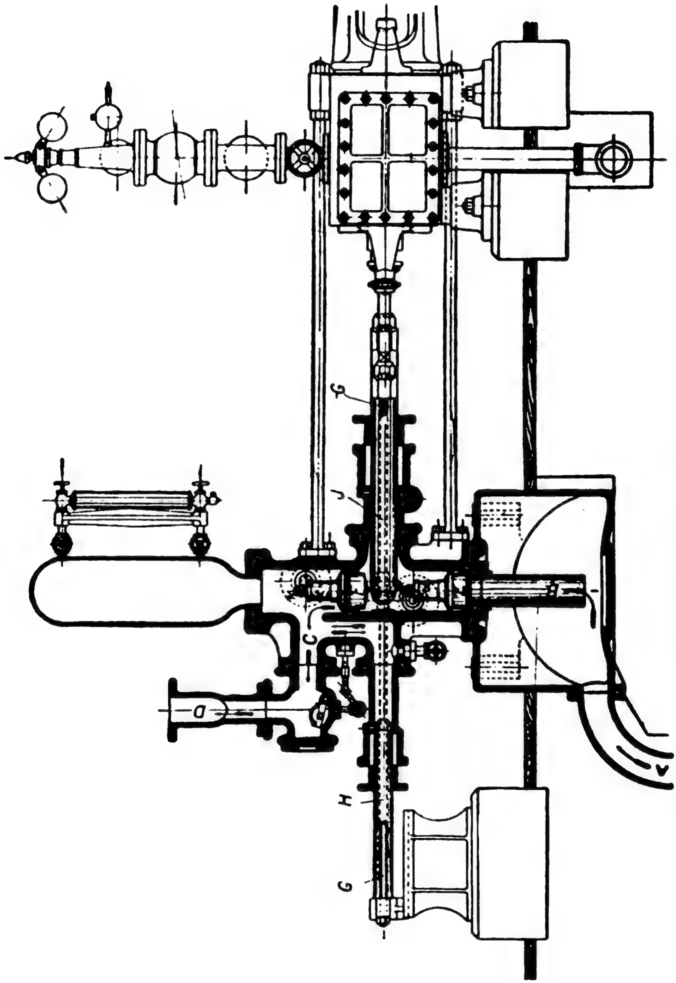


FIG. 397.—Riedler Differential pump.

The cross-sectional areas of the main and differential plungers are generally made in the proportion of about two to one, so as to equalise the work done, as is the case in a double-acting pump. The rods G are the side rods connecting the cross-heads of the main and

differential plungers, these rods being always in tension. In front of C, and connecting to the main pipe D, is a clack valve (shown open). The valve is for the purpose of preventing the water in the pipe from running out when it is desired to remove the valves or examine the interior parts of the pump. This valve is kept open when the pump is working, but can be shut when required by means of levers on the outside of the clack chamber.

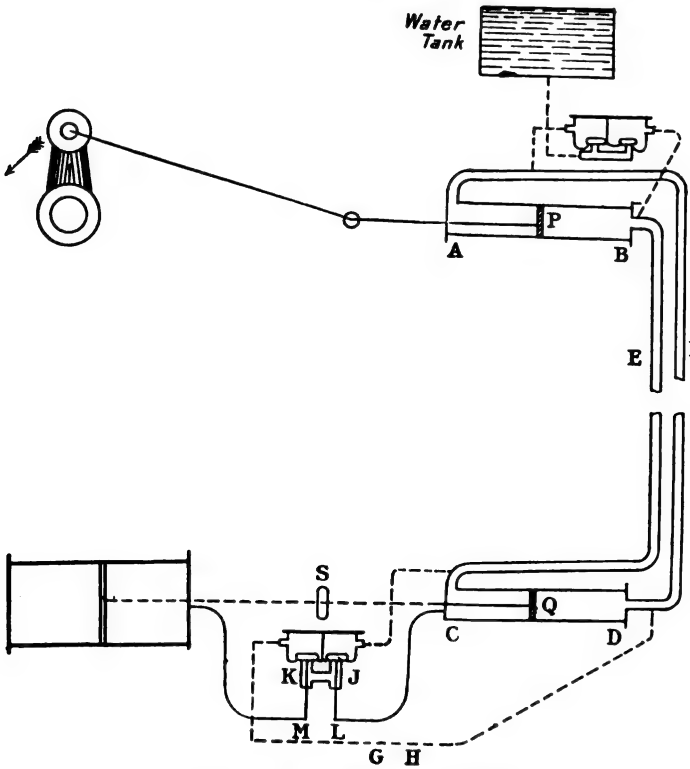


FIG. 398.—Moore's hydraulic pump.

Moore's Hydraulic Pump.—In this pump no rods are used, two columns of water being substituted for the ordinary solid rods connecting the steam engine to the pump. The action of the pump will be understood from fig. 398. On the surface is a cylinder A B, in which travels a piston P, driven by the tumbling crank of a steam engine. Underground there is another cylinder C D, exactly similar to the first. The piston Q, which travels in it, is connected to an ordinary double-acting pump. There are pipes F from the ends A

of the first cylinder A B to the end D of the second C D, and another, E, from the end B to the end C. The pipes and the cylinders are both kept full of water. When the surface piston makes its first stroke, the water is forced out of one end of the cylinder through the pipe to the corresponding end of the cylinder underground, and the piston is then driven back from C to D. When the stroke on the surface is reversed, the piston underground is forced in the opposite direction, and the motion is thus transferred from the engine on the surface to the pump underground, in the same way as would have been done had there been two rods instead of two columns of water. Should there be any leakage in one of these pipes, the plunger at the bottom would make a shorter stroke in the one direction than in the other, and would work towards the end, and unless there was some regulator, would knock the end of the pump off. This is obviated by having valves work by tappets, set at such a distance that, when the stroke is completed, they are opened, and the water allowed to pass from one pipe to the other, thus adjusting the stroke of the rams.

Curves based on the work done by one of these pumps 10 in. diameter, showed an efficiency of 66·26 per cent. The energy lost in the friction of the engine gearing and surface power rams was 10·24 per cent.; in transmitting the power through the pipes 14·36 per cent., and in the friction of the underground rams 9·12 per cent. These figures compare favourably with the efficiency of pumps operated by electricity or compressed air.

This hydraulic pump can work with a column of water of 150 fms. or less, but more satisfactory results are got when the column is between 40 to 80 fms. When it is too great, much trouble may ensue, and the pipe joints become very difficult to keep tight, while the pipes themselves are apt to burst, especially at bends. Again, at long distances (say over a mile underground) these pumps do not work satisfactorily, and in such cases it is better to employ either electricity or compressed air as a motive force.

Kaselowsky Pump.—With this pump a combination of hydraulic pressure and compressed air is used. The system includes at the surface: (1) a steam engine M (fig. 399) operating the water-compression pumps P and the air-compressor C, the latter furnishing the air for the pressure regulator R. Underground the plant consists of: (*a*) an hydraulic motor K, of a special form, operating the lifting pumps G; (*b*) two pressure regulators, R^1 and R^2, for the water pipes.

In the shaft: (*a*) The pipe 1, conducting the water to the pumps; (*b*) the pipe 2, carrying the water discharged or exhausted from the pump; (*c*) the pipe 3, carrying compressed air to the regulators; (*d*) the pipe 4, for the water lifted or discharged from the mine. The water operating the pumps passes from the pump P, through the accumulator A, which regulates the pressure through the pipe 1 to

the motor K at the bottom of the shaft. After performing the work it is returned to the surface through the pipe 2 on which is the regulator R^1, and is discharged into the reservoir S, from which the pump P^1 is supplied. The regulator, as shown, is formed of two cylinders, C_1 and C_2, in which are two pistons, P_1 and P_2, of such diameters that the ratio between them is 1 : 10. The water acting on P, with a pressure, say, of 200 atmospheres (3000 lbs. per sq. in.) is counterbalanced by a compressed air pressure at 25 atmos-

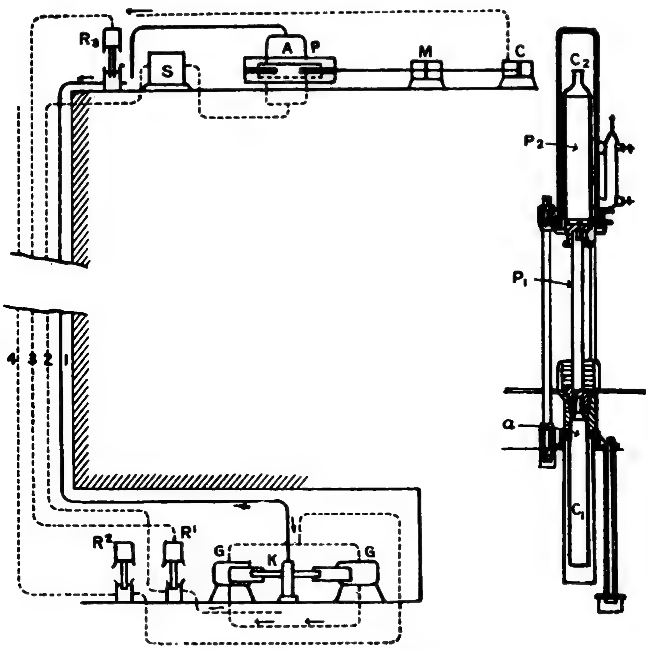


FIG. 399.—Kaselowsky pump.

pheres (375 lbs. per sq. in.) or P_2. The piston is a hollow cylinder of phosphor bronze. The water piston P_1 has a small cavity *a*, which permits the escape of water in case the pressure rises too high and causes the piston P_2 to rise too high. Several pumping plants on this system have been installed at collieries in Germany, where they have given a useful effect of 69 to 75 per cent. The principal advantages of this method are the small size of the different parts, both fixed and moving, and the ability of the pump to work when submerged, which is impossible with steam. On the other hand, it is rather an expensive system of pumping.

Pulsometer Pump.—The Pulsometer is very largely used for pumping water under various conditions, particularly when the lift is small. It will work best, and give the greatest efficiency, when the height to which the water is to be raised is between 30 and 50 ft. It is very suitable for drainage of dip workings, or sinking shafts, or

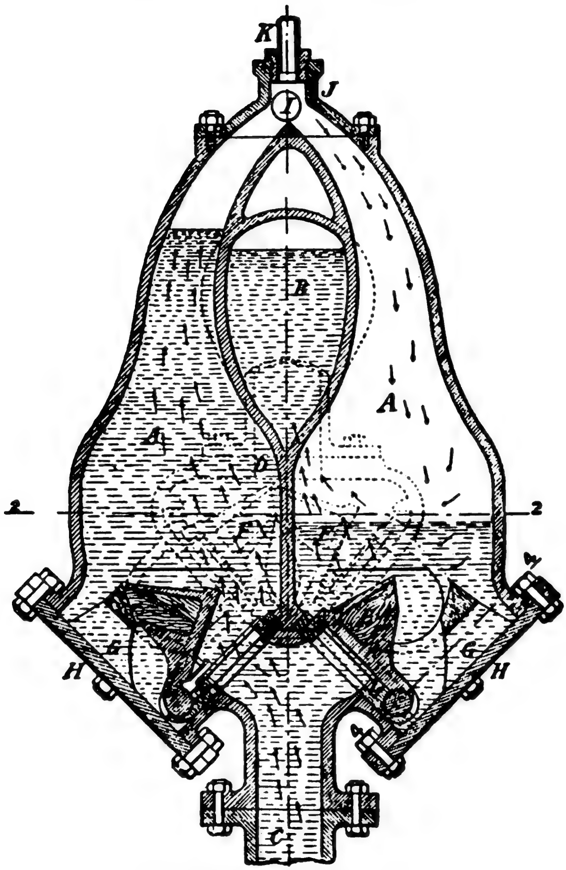


Fig. 400.—Pulsometer pump.

for raising water from settling ponds to coal-washing machines, as it can work fairly well with dirty or gritty water. The pulsometer is entirely different in construction from an ordinary steam pump, inasmuch as it has no steam cylinder, piston, piston-rod, or bucket. Its construction may be understood from the accompanying illustration in fig. 400.

The body consists of a casting shaped somewhat like a pear, and divided into two chambers A A joined side by side, and with tapering necks bent towards each other, surmounted by another casting called the neck J, accurately fitted and bolted to it, in which the two passages terminate in a common chamber, wherein is fitted the ball-valve I, which can oscillate between seats formed at the junction of the neck. Downwards, the chambers A A are connected with the suction passage C, wherein the inlet or suction valves E E are arranged. A discharge chamber, common to both chambers, and leading to the discharge pipe, is also provided, and this also contains one or two valves F F, according to the purpose for which the pump is required. The air-chamber B communicates with the suction. The suction and discharge chambers are closed by covers H H, accurately fitted to the outlets. These can be readily removed when access to the valves is required.

Starting the Pulsometer.—To set it at work, the pump is filled with water, either by pouring water through the plug-hole in the chamber, or by drawing the discharge. Steam being admitted through the pipe K, by opening the stop-valve to a small extent, it passes down that side of the neck which is left open to it by the position of the ball, and presses upon the small surface of water in the chamber which is exposed to it, depressing it without agitation, and consequently with very little condensation, and driving it through the discharge opening and valve into the rising main.

The moment that the level of the water is low enough to uncover the horizontal orifice which leads to the discharge, the steam blows through with a certain amount of violence, and being brought into intimate contact with the water in the pipes leading to the discharge chamber, *instantaneous condensation takes place*, and a vacuum is in consequence so rapidly formed in the newly emptied chamber that the steam ball is pulled over into the seat opposite to that which it occupied during the emptying of the chamber, closing its upper orifice and preventing the further admission of steam, and making the vacuum complete until water rushes in, as it does immediately through the suction pipe, lifting the inlet valve E and rapidly filling the chamber A again.

The condition of things is now exactly in the same state in the second chamber as it was in the first, and similar effects are therefore obtained.

Small air-cocks are screwed into the cylinders and air-chambers, to prevent the too rapid filling of the chambers on low lifts. While the pulsometer is admittedly a handy and useful pump under various conditions, it has the drawback that it consumes a large amount of steam for the work done, compared with direct-acting steam pumps. For instance, a pulsometer raising 100 gallons of water per minute, to a height of only 26·25 ft., required 29·7 lbs. of coal per H.P. per hour, which is a very large consumption, considering the conditions.

A good deal of difficulty is also sometimes experienced with the steam ball valve getting worn flat in some places and sticking.

Centrifugal Pumps.—Centrifugal pumps have been long used as blowers for air in forges and furnaces, and are now largely used for raising water to moderate elevations. They are particularly well adapted for disposing of dirty gritty water, such as the discharge water from coal-washing machines, or where the water from machines has to be used over again, providing the height be not too great. They will work under such conditions much more efficiently than pulsometers. The construction of the pump will be understood from fig. 401. Inside a flat casing of approximately circular outline are the paddles or blades, which extend from near the centre outwards to the circumference, and are usually curved backwards. The water between the blades tends, in virtue of the centrifugal force, to move outwards, and is allowed to pass off through a large discharge orifice tangential to the circle described by the paddles. The height to which water can be raised, if there is no loss, by centrifugal pumps, may be expressed by the formula $h = \frac{v^2}{2g}$; in practice, however, h is only equal to about $\frac{3}{4}\frac{v^2}{2g}$. The velocity of the blades is usually taken at $N = 10\sqrt{h}$.

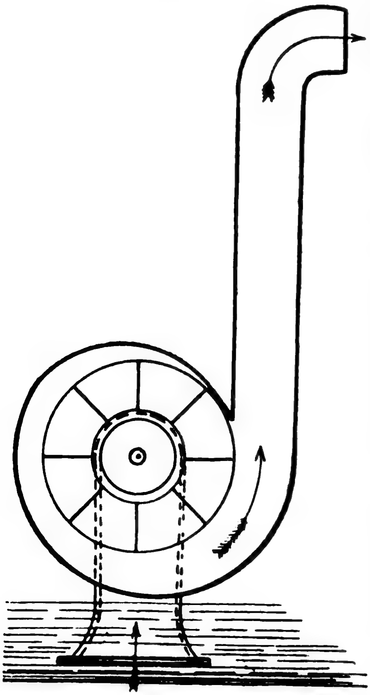


FIG. 401.—Centrifugal pump.

Centrifugal pumps, to work well, must have a very short suction pipe, or, what is better, have the water flowing into them, or be submerged altogether in the water to be raised.

If not submerged, they ought to be primed with water before being started, otherwise, in driving the air out of them, they simply act as a blower.

With the type of centrifugal pump just described only a very limited head could be dealt with, 50 to 100 ft., and even with this a low efficiency was generally the result. The problem was to get a centrifugal pump that would pump water against a head of a greater

height than 100 ft., say 500 to 1000 ft. This has now been accomplished by running centrifugal pumps in series. Thus if an ordinary single pump can deliver water against a head of 40 ft., the addition of another chamber will give a final delivery of 80 ft., while three chambers will enable the pump to discharge the same quantity of water against a head of 120 ft. In collieries where frequently large quantities of water, often dirty and gritty, have to be dealt with, such pumps are of great utility. The writer recently saw centrifugal pumps, of the Gywnne type, dealing with 500 to 600 gallons of water per minute against a head of nearly 300 ft., and they gave very little trouble. Messrs Mather & Platt, of Manchester, have recently introduced a high-lift centrifugal pump, capable of discharging water against a head varying from 250 to 500 ft.; several of these pumps being at work in Scotch collieries. This machine (fig. 402) is known as the 'Patent High-Lift Turbine Pump,' and its main feature is that it consists of one or more sets of vanes, or impellers, each set running in its own chamber, but upon a common shaft, the delivery pressure of the water varying directly as the number of chambers used. In Mather & Platt's patent centrifugal pump the water enters the revolving wheel axially, traverses the curved internal passages between the vanes, and is discharged tangentially at the periphery into a guide ring of special construction; this conveys it to the annular chamber in the body of the pump, where the velocity head imparted to the water by the wheel is converted into pressure head. From this chamber the water is finally discharged into the delivery pipes, or, if the pump be a multiple one, into the second and subsequent chambers. A special feature of this pump is the provision of the stationary guide ring mentioned above; this is fixed concentric with the revolving vanes, and, owing to its design, enables the conversion of velocity into pressure head more perfectly than hitherto, thus increasing the possible height of lift and efficiency of the pump. There can be no doubt that pumps of the types described have advantages over the ordinary plunger pumps so largely used in mining work, and they are specially useful for dip workings. They can be easily moved into a new position with the extension of the workings, which in itself is a great advantage. Other advantages claimed for these centrifugal pumps are:—

They have few moving parts in contact, thus reducing wear and tear.

As they can be designed to run at a high speed, they occupy little space.

Heavy foundations are unnecessary, as even large pumps can be fixed to wooden beams or fixed on a moving bogie.

They can deal with dirty and gritty water more efficiently than the ordinary type of steam pump.

The following formulæ are used for calculations with centrifugal pumps:—

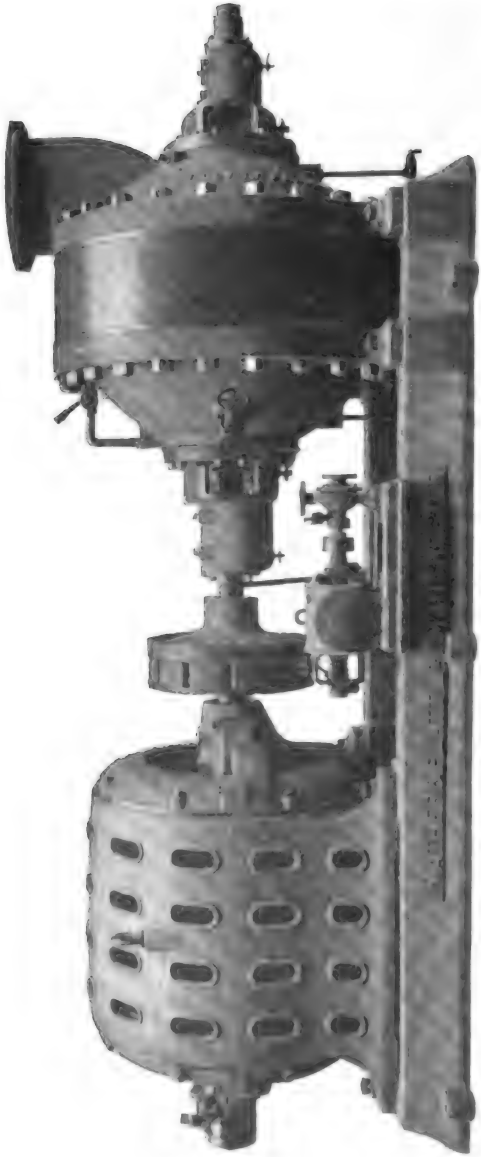


FIG. 402.—Mather & Platt high-lift turbine pump.

(I.) *To find the required peripheral speed of the impeller or wheel in feet per second for a given head.*

$$V = k\sqrt{H}$$

Where V = peripheral speed in feet per second.
H = head water is to be delivered against in feet.
k = a coefficient = 8 for small pumps and 9·82 for large pumps.

(II.) *To find diameter of wheel for a given quantity of water and given head.*

$$D = \sqrt{\frac{Q}{\sqrt{H} \times 0{\cdot}16}}$$

where D = diameter of wheel in feet.
Q = quantity of water in cubic feet per minute.
H = head in feet.
0·16 = coefficient.

(III.) *To find revolutions of wheel per minute, when height of delivery and circumference of wheel are given.*

$$R = \frac{\{(8\sqrt{H}) \times k\} \times 60}{Q}$$

$$\text{or } R = k\frac{\sqrt{H}}{D}$$

R = revolutions per minute.
H and Q = same value as above.
k = coefficient as in (I.)

k in this case being = 153 for small pumps and 187 for large pumps.

(IV.) *To find effective horse-power required, given gallons per minute, for a given quantity of water to be delivered against a given head.*

$$\text{E.H.P.} = \frac{G \times 10 \times H \times {\cdot}66}{33{,}000}$$

E.H.P. = effective horse power.
G = gallons to be delivered per minute.
H = same value as above.

(V.) *To find diameter of suction and delivery pipes in inches.*

$$d = {\cdot}225\sqrt{g}$$

d = diameter of pipe in inches.
g = gallons delivered per minute.

Sinking Pump.—During sinking operations an arrangement of pumps is required differing from that in use under ordinary circumstances. A common arrangement in sinking is a 'sliding suction' pump working through a 'packing gland.' It may be lowered as the sinking proceeds. A short joining piece is used between the sliding suction and the working barrel, this short piece being made of weaker metal than the other parts, so that in case of a side stroke from a shot it may give way, without injuring the more expensive parts. The method of operating is usually to fix all the pipes above the working barrel with collaring, and to allow the working barrel and sliding piece to be lowered as sinking proceeds. The sliding piece is generally made the length of one of the pipes, so that when it has been let out this distance, the column is 'cut' above the working barrel and another length added, when the pumping may proceed as before. The objection to this method is, that the pipe column requires to be cut at intervals, say 9 ft., as the sinking proceeds; but if the pumps are not large it works very satisfactorily.

For light pumps a strong flexible hose may be used instead of the sliding suction. The pump rods are usually longer than the column of pipes, and short pieces are used as lengthening parts, or an arrangement with a gland fixed to the bell-crank, and the rods clamped to it, is used. This is more satisfactory, as the rods can then be put in the full length and lengthened as sinking proceeds.

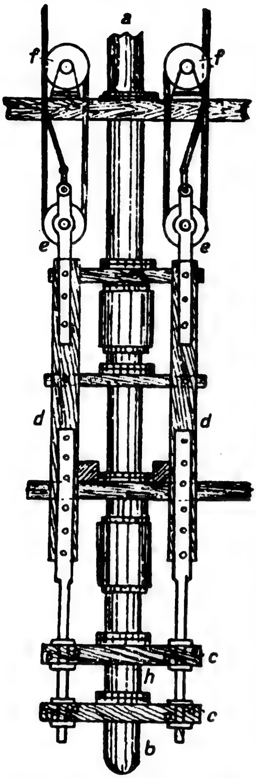


Fig. 403.—Sinking pump with lowering gear.

The second method of employing a sinking pump is to lower the whole column, either with iron rods and screws, or by ropes worked from a steam winch at the surface, as the work proceeds. A combination of both of these methods is possibly the best and safest. Fig. 403 shows the arrangement of lowering a sinking set with ropes and ground spears. The pipe column *a* is fixed rigidly to the suction piece *b*, and between the suction piece and the clack piece a short pipe *h* is inserted, having extra broad and strong flanges. Immediately below the flanges, two strong iron glands *c c* are fixed and connected to the 'ground spears' *d d*; at the top of these spears are two sheaves *e e*, connected to the spears with strapping plates. At the surface two similar sheaves *f f* are fixed, round which the ropes *g g* work. They are operated by a steam winch at the surface, so that the whole lift can be lowered as sinking proceeds, and fresh pipes added at the top as required. The spears may be made of sections of pitch pine 4 in. to 8 in. square, according to the size of lift, or they may be of wrought iron 2 in. to 4 in. diameter. They ought to be well strengthened at frequent intervals with cross glands.

A third arrangement is to raise the water, while sinking proceeds, by means of a steam pump slung in the shaft, and connected to lowering screws of wrought iron which work through beams at the surface, with a large nut operated by spanners. As an additional precaution, a strong rope, operated by a steam winch at the surface, should also be connected to the pump, it being likewise useful for lowering the pump or other parts when pieces are being added to the screws. This method of pumping has much to commend it, as it does away with the necessity for using pump-rods and bell-cranks. The only difficulty is that a good deal of space is required if the

quantity of water to be dealt with is large, as the engine has to be correspondingly large. Good strong tackle must be used, as the weight of pump and connections alone often amounts to between 12 and 18 tons. In this method the whole contrivance may be lowered as sinking proceeds, or the pipes may be fixed and the pump lowered alone, the pipes immediately above it being cut as required. This latter method is probably the safest and best in most cases. The usual method of arranging the pumps in the shaft is to make the sinking or bottom lift a bucket lift, until a point has been reached where it is proposed to put in a permanent lift. A plunger lift may then be put in, and the sinking can proceed as before, with the bucket set. In some instances plunger pumps are used until the lowest part of the sinking is reached, when a bucket lift is substituted with which to complete the sinking. The great advantage of the bucket lift is that, as has been seen, it requires very little space in the shaft, as the rods work inside the pipes.

The arrangement of sinking pumps adopted by Messrs Joseph Evans & Sons of Wolverhampton is a very suitable and handy method of dealing with water in sinking pits, and has been found to work satisfactorily.

The pump itself (see fig. 404) is what is known as a vertical 'Cornish' sinking pump, is double-acting, of the outside packed ram type fitted with differential ram, and is suitable for heads up to 300 ft. Pumps of this type were employed at the two shafts recently sunk by the Niddrie and Benhar Coal Co., Ltd., at Olive Bank, Musselburgh, near Edinburgh, where a large quantity of water was met with in the sinkings. The following were the sizes of pumps used at these pits:—two with 14-in. diameter steam cylinders, 9-in. water cylinder with 2 ft. stroke, each capable of delivering 275 gallons per minute at a speed of 100 ft. per minute; one 16-in. diameter steam cylinder, 9-in. water cylinder, and delivering 275 gallons per minute; two 24-in. diameter steam cylinders, 12-in. water cylinders by 2 ft. stroke, each capable of delivering 490 gallons per minute at a speed of 100 ft. per minute. Each pump was required to deliver the water to a height of 300 ft., with a steam pressure of 100 lbs. per square inch at the boilers. The pumps were each fitted with removable lined working barrels, so that in the case of the barrels becoming worn or scored by dirty or gritty water, they could be withdrawn and rebored without taking the pumps out of the shaft. These pumps worked with great smoothness, and were admirably suited for the work they had to do, for in this case it would have been almost impossible to have used pumps operated by the ordinary method of pump-rods and bell-cranks, owing to the uncertain nature of the surface and the subsidence which took place.

Air-Vessels.—Every pump in which the plunger or bucket moves at a greater speed than 40 ft. per minute, and in which the area of the pipes is not larger than that of the plunger or bucket, will work

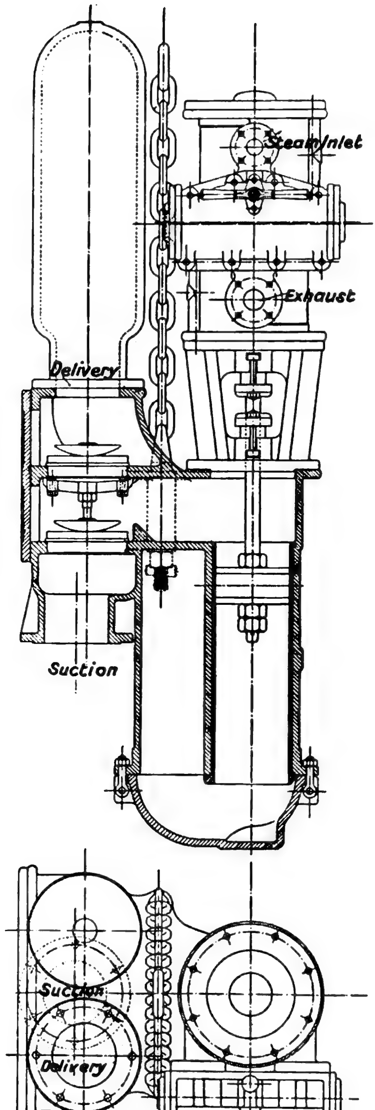


FIG. 404.—Evans' sinking vertical Cornish steam pump, with removable cast-iron liner.

better and more smoothly with an air-vessel than without one. The primary object of an air-vessel is to reduce the shocks that are liable to occur inside the pumps and pipes, especially with plunger pumps, and thereby equalise the pressure necessary to force the water up the delivery pipes. When an air-vessel is connected with the pumps, a greater velocity can be obtained, either with plunger or bucket, with the same degree of safety, thereby increasing the capacity of the pumps.

The benefits to be derived from air-vessels on pumps are much questioned by many engineers, some averring that they are of no service, or that the pump does better without them. If the air-vessel be badly placed, or badly attended to, in not being charged properly, its presence will doubtless be a hindrance to the efficient working of the pump. The nearer the air-vessel is to the casing or barrel the better. If only one is used it should be placed immediately above the discharge clack on the delivery side. In slow-moving pumps, air-vessels are not actually required. To get the maximum benefit from air-vessels they must be well looked after and kept constantly charged with air. In one of the best forms of air-vessel for a vertical column of pipes, the rising column is enlarged and an inside pipe is brought down near to the bottom of the chamber. Immediately below this pipe is fixed a cup, which tends to divert all the particles of air flowing with the water into the chamber.

To charge the air-chamber, the tank is connected to the rising column by a small pipe with a tap to it. At the other end of the tank are two small pipes for ingress and egress. The tank is first filled with water by opening the tap, and keeping the other pipes closed. When it is filled, the tap is closed and the pipes opened, when the water rushes out at and air enters by the pipe and rises from the tank into the air-chamber.

An important point in connection with air-vessels is to secure their being of sufficient area. This should be four to six times the area of the working barrel, according to the speed of pump.

The capacity of air-chamber necessary will depend upon the type of pump used, single-acting pumps requiring much larger chambers than double-acting ones. Chambers made of cast iron should be well tested for tightness under full pressure; they ought also to be provided with pressure-gauge glasses to show the water level, or, if the pressures are high, a series of 'try' cocks should be fixed, for the purpose of ascertaining the position of the air in the chamber.

'Duty' of Pumping Engines.—The duty or efficiency of a pumping engine is measured by the number of foot-pounds of work performed per cwt. of coal consumed. It varies greatly with the type of engine and the circumstances under which it works.

The duty of pumping engines was first recorded in Cornwall by Watt in connection with engines which he erected at some of the

mines there. Naturally, in such a district, where the price of fuel is high, there would be a desire to get the largest amount of work possible for a given coal consumption. The highest efficiency ever obtained was from a Cornish pumping engine which gave 146 million foot-pounds per cwt. of coal burned. After allowing for friction, this corresponds to a consumption of 1·21 lbs. of coal per hour per indicated horse-power, which is very high efficiency indeed, as, with ordinary colliery pumping engines, it is not unusual to find a consumption of 10 to 15 lbs. of coal per horse-power per hour, and in some types of engines a great deal more. The 'duty,' as defined above, includes the efficiency of the boilers and engine, and depends a good deal on the quality of the fuel burned.

A fair comparison of pumping engines may be made, and the efficiency ascertained by testing the consumption of coal of average quality in raising the steam required, and may be calculated from the formula:

$$U = \frac{(H \times (d^2) \times 2{\cdot}045 \times N \times L)}{W},$$

where H is the height of lift of pump in fathoms; d the diameter of pumps in inches; L the length of stroke; N the number of strokes in one month; W the weight of water in lbs.; U the units of work per lb. of steam (duty).

The test to ascertain the efficiency is generally made when the ordinary work of the mine is suspended.

The table gives the result of the 'duty' performed by various pumping engines from actual tests made by Mr Dugald Baird on the foregoing basis.*

Class of Engine.	Position of Engine.	Diameter of Cylinder in in.	Length of Stroke.	Depth of Lift in Yds.	Duty.
a. Bull	Surface	100	12 ft.	300	50,265,312
b. Davey differential	Underground	36	4 ,,	300	41,360,000
c. Davey differential-compound	Surface	33-in. H.P.C. / 52 ,, L.P.C.	9 ,, / 12 ,,	200	69,705,890
d. Duplex differential	Underground	22	2 ,,	300	41,360,000
e. Compound Bull	,,	28 ,, H.P.C. / 48 ,, L.P.C.	12 ,,	168	35,827,200
f. Compound Davey differential	..	34 ,, H.P.C. / 64 ,, L.P.C.	7 ,, 2¼ in.	..	100,800,000

a, *b* and *d*, Leven Colliery; *c* and *f*, Davey's; *e*, Wellsgreen.

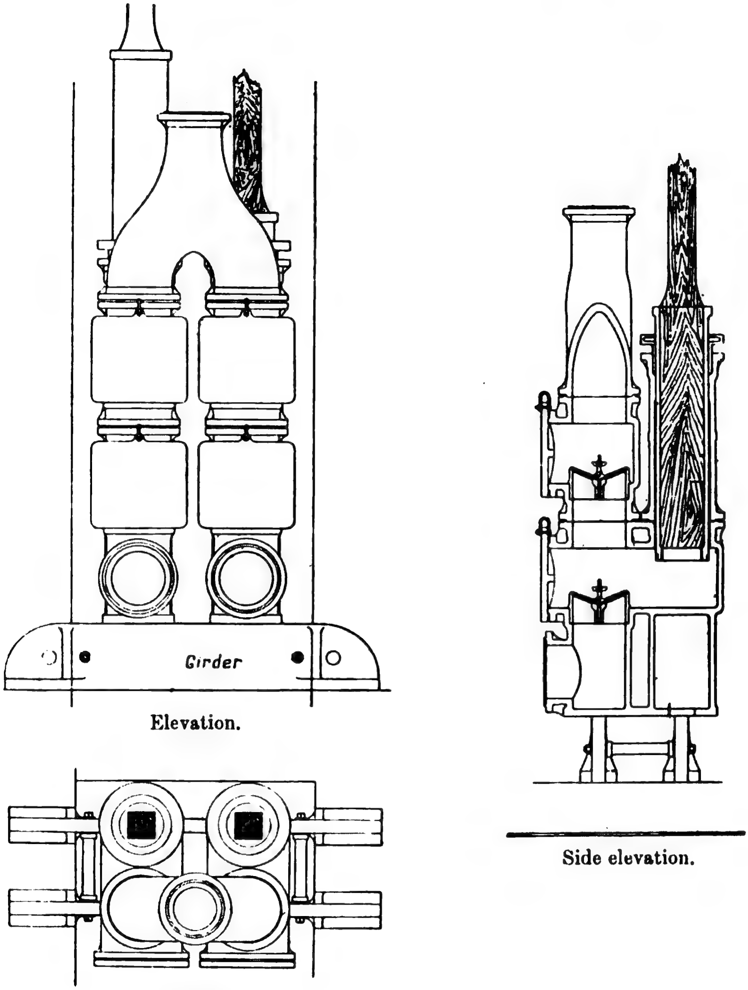
Mr Henry Davey states that the greatest efficiency of double-acting rotative pumping engines does not exceed 30 to 40 million foot-lbs. per cwt. of coal burned.

Arrangement of Pumps.—As already stated, it is usual to make the bottom lift in the shaft a bucket pump, and the other lifts forcing or plunger sets, which arrangement is found to work satisfactorily in most cases.

Many, however, prefer to have all the lifts as forcing sets, especially

* *Trans. F. Inst. Min. Eng.*, vol. xi. p. 100.

if there is no likelihood of the lower lift being flooded by a sudden inflow. Where pumps are operated with rods in the shaft, it is



Elevation.

Side elevation.

Plan.

FIGS. 405, 406, 407.—Double plunger pump.

almost an invariable custom at coal mines to work the rods with a pair of bell-cranks, unless they are worked by a direct-acting engine, such as a Bull engine. By using double bell-cranks the power is more

evenly distributed, and where a number of lifts have to be operated, fewer offsets are required than if the pumps were all worked off a single bell-crank. Figs. 406, 407 show front and side elevations of a good arrangement for working two or more lifts by a pair of bell-cranks. The features which commend this arrangement are that the pump is a double-acting one, and that there are two distinct sets of rods carried down from the surface, the lower lift in each case being worked by a cross-head on the rods of the lift above.

When the sets are large, *i.e.* above 20 in. diameter, it is often a difficult matter to fix on a suitable arrangement to occupy as little space as possible. Figs. 405, 406, 407 show a plan, elevation, and side elevation, respectively, of a very compact arrangement for a double plunger set of 24 in. diameter, with one central delivery column common to both pumps. In this disposition of the pumps as little space as possible is taken up, and it is in every way convenient. The foundations for the pumps are generally strong beams of timber built into the shaft, or wrought-iron or steel girders, the latter being much to be preferred, as they give the maximum of strength with a minimum of space, and are less liable to decay than timber supports.

A few worked-out examples on pumping, such as are often set at examinations, are given below, in the hope that they may prove useful.

Question.—How many strokes per minute can be made by the piston of a pump whose area is 2 sq. ft., length of stroke 5 ft., and the height to which the water is raised 60 fms., driven by an engine of 80 horse-power?

$$\text{H.P.} \times 33000 = A \times L \times 62{\cdot}5 \times d \times x.$$

When H.P. = horse power, A = area of pump in sq. ft., L = length of stroke in feet, 62·5 = number of lbs. in 1 cub. ft. of water, d = height in feet water is raised, x = the number of strokes required per minute.

$$\therefore\ 80 \times 33000 = 2 \times 5 \times 62{\cdot}5 \times 360 \times x$$

after cancelling $880 = 75x \quad \therefore\ x = \dfrac{880}{75} = 11{\cdot}86.$

Question.—At what rate will it be necessary to work a pump 12 in. diameter with 4½ ft. stroke, to deal with 200 gallons of water per minute in a shaft 200 yds. deep, and what is the approximate horse-power required?

Using the same letters as above, and G = gallons of water per minute, and D = diameter of pump in inches, 6·25 = number of gallons in 1 cub. ft.

Allowing 10 per cent. of loss for slip, $G = 200 + 20 = 220$.

$\therefore\ G = \dfrac{6{\cdot}25 \times D^2 \times {\cdot}7854 \times L \times x}{114}$, or calculation can be simplified, as $\dfrac{6{\cdot}25 \times {\cdot}7854}{144}$

$$= {\cdot}034 \quad \therefore\ G = D^2 \times {\cdot}034 \times L \times x$$
$$220 = 12^2 \times {\cdot}034 \times 4{\cdot}5 \times x$$
$$220 = 22{\cdot}03\ x$$
$$\therefore\ x = \frac{220}{22{\cdot}03} = 9{\cdot}16 \text{ strokes per minute.}$$

As 1 gallon of water is equal to 10 lbs. $\therefore$ H.P. $\times 33000 = 220 \times 10 \times 200 \times 3$

$$\text{or H.P.} = \frac{220 \times 10 \times 200 \times 3}{33000} = 40$$

Question.—A hydraulic pump having an 8-in. diameter plunger is wrought by means of a head of water brought from the surface in pipes 2 in diameter. Find the total pressure on plunger and weight of water in pipes, if the depth of the shaft is 360 ft.

Pressure in lbs. per sq. in. due to head of water $= \frac{d \times 12 \times 62{\cdot}5}{1728}$, or as $\frac{12 \times 62{\cdot}5}{1728}$ $= {\cdot}434$ $\therefore$ pressure in lbs. per sq. in. due to head of water $= d \times {\cdot}434 = 360 \times {\cdot}434$ $= 156{\cdot}24$ lbs.

Total pressure on plunger = area of plunger in sq. in. $\times$ pressure in lbs. per sq. in. $= 8^2 \times {\cdot}7854 \times 156{\cdot}24 = 7852{\cdot}62$ lbs.

$$\text{Weight of water in pipes} \begin{cases} = 156{\cdot}24 \times 2 \times {\cdot}7854 \\ = 490{\cdot}51 \text{ lbs.} \end{cases}$$

Question.—What number of gallons and cubic feet of water can be pumped per hour from a pit 600 feet deep, by an engine of 200 H.P., assuming the efficiency of the engine to be ·6?

Let x = weight of water in lbs. raised per hour.

$$\text{Then } x \times d = \text{H.P.} \times 33000 \times 60 \times 6$$
$$x \times 600 = 200 \times 33000 \times 60 \times {\cdot}6$$
$$\text{or } x = \frac{200 \times 33000 \times 60 \times {\cdot}6}{600}$$
$$= 396000 \text{ lbs.}$$

$$1 \text{ gallon of water} = 10 \text{ lbs.} \quad \therefore \text{gallons per hour} = \frac{396000}{10} = 39600$$

$$1 \text{ cub. ft. of water} = 62{\cdot}5 \text{ lbs.} \quad \therefore \text{cub. ft. per hour} = \frac{396000}{62{\cdot}5} = 6336$$

Question.—Find the quantity of water delivered by a double-acting plunger pump, if the plunger is 7 in. in diameter, length of stroke $4\frac{1}{2}$ ft., and working at 20 strokes per minute. Also find the horse-power if the shaft be 90 fms. deep.

Using the same notation as in Question 2, and also let N = number of strokes of plunger per minute,

$$\text{Then } G = D^2 \times {\cdot}034 \times L \times 2 \times N$$
$$= 7^2 \times {\cdot}034 \times 4{\cdot}5 \times 2 \times 20$$
$$= 299{\cdot}88 \text{ gallons per minute.}$$
$$\text{H.P.} \times 33000 = G \times 10 \times d$$
$$\text{H.P.} \times 33000 = 299{\cdot}88 \times 10 \times 90 \times 6.$$

$$\text{After cancelling H.P.} = \frac{299{\cdot}88 \times 9}{55} = 49{\cdot}07.$$

Question.—Give the principal sizes of a direct-acting pumping engine fitted with a fly-wheel which you would erect to raise 400 gallons of water per minute from a depth of 80 fms.; allowing $\frac{1}{3}$ for stoppages, $\frac{2}{3}$ for efficiency of engine, and 10 per cent. for slip of pumps.

As $\frac{1}{3}$ has to be allowed for stoppages, and 10 per cent. for slip of pumps,

$\therefore$ Gallons to be raised per minute $= \frac{400 \times 24}{16} + 10$ per cent. $= 660$.

We would therefore require to provide a pump capable of raising 660 gallons per minute. Assume the speed of the pump to be 100 ft. per minute.

$$G = D^2 \times \cdot 034 \times \text{speed}$$

or $D = \sqrt{\frac{\text{gallons per minute}}{\cdot 034 \times \text{speed}}} = \sqrt{\frac{660}{\cdot 034 \times 100}} = 13 \cdot 9$, or 14 in. approximately as the diameter of pump required.

Then to calculate size of engine required to raise 600 gallons per minute, we may equate the work thus:—

Work done by engine = work done in shaft;
or $D^2 \times \cdot 7854 \times P \times \text{speed} \times E =$ weight of water in lbs. per minute $\times$ height to be raised in feet.
Where $E =$ efficiency of engine $= \frac{2}{3}$ or $\cdot 66$.

If we assume the effective steam pressure to be 50 lbs. per sq. in., and the speed of engine to be the same as the pump, 100 ft. per minute,

Then $D^2 \times \cdot 7854 \times 50 \times 100 \times \cdot 66 = 600 \times 10 \times 80 \times 6$,
after cancelling $\cdot 08639\ D^2 = 96$

and $D = \sqrt{\frac{96}{\cdot 08639}} = 33 \cdot 3$ in. diameter of cylinder.

The engine is to be direct-acting, so that it could have a 5 ft. stroke, and run at the rate of ten double strokes per minute; the diameter of cylinder being 33 in., and the steam pressure 50 lbs. per square in. Suppose the above engine to be compound and working expansively, and the steam to be cut off at $\frac{1}{3}$ of the stroke; what would the diameter of the low-pressure cylinder require to be?

The size of the low-pressure cylinder may be found by the formula:

$a = \frac{A}{\sqrt{E}}$: Where $a =$ area in sq. in. of high-pressure cylinder

$A =$,, ,, low ,, ,,

$E =$ number of expansions of steam in cylinder $= \frac{P}{T}$

$P =$ mean effective steam pressure in lbs. per sq. in.

$T =$ terminal pressure of steam $= P\frac{l}{l_1}$

$l =$ length of stroke before steam is cut off

$l_1 =$,, ,, after ,, ,,

Here $T = 50 \times \frac{20}{60} = 16{\cdot}6$ lbs., and $E = \frac{50}{16{\cdot}6} = 3$, and by formula $a = \frac{A}{\sqrt{E}}$, or if we let d = diameter of high pressure cylinder
and D = „ low „ „

$$\text{then } d^2 \times {\cdot}7854 = \frac{D^2 \times {\cdot}7854}{\sqrt{E}}$$

$$\therefore D^2 \times {\cdot}7854 = d^2 \times {\cdot}7854 \times \sqrt{E}$$

$$D = \sqrt{\frac{33^2 \times {\cdot}7854 \times \sqrt{3}}{{\cdot}7854}} = \sqrt{1883{\cdot}97} = 43{\cdot}4 \text{ in.}$$

If we allow the same efficiency for this cylinder as for the high-pressure one, then its diameter would require to be $43{\cdot}4 + \frac{1}{3} = 57{\cdot}8$ in. The ratio between the two cylinders is often taken as 1 : 1·6 or 1·5 when the number of expansions is less than 10.

CHAPTER XIII.

VENTILATION.

Gases Present in Mines.—The principal causes of impure air in mines are:—The exhalations of men and animals; burning lamps or candles; gases given off naturally from the strata and those resulting from blasting; decaying timber in the workings; absorption of oxygen by chemical agencies; introduction of foreign substances.

In breathing, oxygen is withdrawn from the air and CO_2 is given off, together with a certain percentage of nitrogen; a man working for eight hours will give off, on an average, over 5 cub. ft. of CO_2. The quantity of air inhaled by a man is said to be ·42 to ·45 cub. ft. per minute, or 25·2 to 27 cub. ft. per hour when at rest; but in a mine it will require much more than this to keep the air pure, possibly 100 to 120 cub. ft. per minute in a non-fiery mine, and in fiery mines, where much gas is given off, 300 to 400 cub. ft. per minute, while every horse in the mine will require three to six times as much air as a man. The quantity of air supplied to a colliery should bear some ratio to the amount of coal raised per shift, because the more actively the working proceeds the greater will be the amount of gas liberated. In non-fiery pits 80 to 100 cub. ft. of air per minute per ton of coal raised should suffice, and 100 to 200 cub. ft. per minute per ton of coal raised in fiery collieries. No hard and fast line can be drawn that will suit every case.

Burning lamps or candles also absorb the oxygen from the air and give off deleterious gases, principally CO_2 and small quantities of CO.

Fire-damp and choke-damp are given off more or less freely in nearly all mines, and are often the principal causes of impure air in the workings.

Timber in some mines, especially in return airways, decays very rapidly and pollutes the atmosphere. To prevent this as much as possible, the bark should be peeled off and the timber thoroughly seasoned before being used underground.

Chemical agencies, such as the action of water on iron pyrites or other ferruginous minerals, absorb considerable quantities of oxygen,

and give off H_2S. The coal itself absorbs the oxyg d by such an extent as sometimes to ignite spontaneously.

The introduction of foreign substances into the air blasting by explosives, the gases naturally given off and fine coal-dust held in suspension, produced by break coal, or carried down into the workings with the intake from the screens on the surface. The statement by ma that explosives cause no fumes is wrong, as nearly all explos off CO and CO_2 in varying quantities, and also a considerable unt of solid residue. The smoke of gunpowder is largely composed of fine particles of carbonate and sulphide of potassium. Dynamite, when exploded, sends into the air, in a finely divided state, the 25 per cent. of infusorial earth which it contains. Boring shot-holes either in the rock or coal also sets in motion considerable quantities of fine dust which help to pollute the air.

From these causes, the atmosphere in the workings is very soon rendered impure and dangerous to breathe, unless means are adopted to clear them, and keep men and animals supplied with pure air. The only safe and practical method is to keep up a current of air of sufficient quantity and velocity as to carry off the deleterious gases as soon as formed.

As the symbols, specific gravities, and atomic or relative weights of the gases met with in mines are frequently referred to, it will be as well to commence by defining these terms.

The Specific Gravity or Density of a body is the ratio of the quantity of matter contained in a given volume to the quantity of matter contained in an equal volume of a substance chosen as a standard. Air is nearly always the standard adopted when comparing the specific gravity of gases.

The Atomic Weight is the lowest proportion by weight of an element which can combine *chemically* with one part, by weight, of hydrogen.

Symbols.—The chemist divides all substances into elements, compounds, and mixtures. Of the former there are between 60 and 70, and for the sake of convenience and brevity in referring to them, the first letter only, or two distinctive letters of the names, are used. Thus H is the symbol for hydrogen, O for oxygen, etc., etc.

Gases are divided into three classes, viz., elementary or simple gases, compound gases, mechanical mixtures. An elementary or simple gas consists of one element only, *i.e.* of a substance which it is impossible to split up or divide.

Compound gases are composed of two or more elements chemically combined with each other. This combination results in the production of a gas differing in its properties from either of the elements of which it is composed.

Mechanical mixture takes place when two or more substances or elements are brought together and no chemical action results. The

atmosphere of a mine is a mechanical mixture of air and the various gases and emanations described above.

The elementary or simple substances of which the compound gases found in mines are composed, are—Hydrogen, H; Oxygen, O; Nitrogen, N; Carbon, C; Sulphur, S.

Hydrogen.—Symbol, H; atomic weight, 1; density 0·0693 (air = 1). An inflammable gas; possessing, when pure, neither colour, taste, nor smell; and a non-supporter of combustion or life. The fact may here be noted that all inflammable gases are non-supporters of combustion in the ordinary sense. Hydrogen being the lightest substance known, it is usually taken as the standard of atomic weight, the weight of all other gases being expressed in terms of hydrogen as unity. 1000 cub. ft. of hydrogen at 14·7 lbs. (atmospheric) pressure per sq. in., and at a temperature of 32° F., weigh 5·606 lbs.

Oxygen.—Symbol, O; atomic weight, 16. Oxygen occurs in the free state in the atmosphere, mechanically mixed with about four times its volume of the inert gas, nitrogen, which acts as a diluent to the highly active oxygen. Oxygen has neither colour, taste, nor smell; does not burn in air, but is the great supporter of combustion and life. All forms of burning, breathing, decay, etc., are simply manifestations of the combination of various substances with oxygen. Since 1000 cub. ft. of hydrogen weigh 5·606 lbs., 1000 cub. ft. of oxygen $= 5·606 \times 16 = 89·69$ lbs.

Nitrogen.—Symbol, N; atomic weight, 14; non-inflammable gas; no colour, taste, or smell, and does not support combustion. It forms $\frac{4}{5}$ths by volume of the atmosphere, but is altogether a very inert gas, being very inactive in all its qualities under ordinary conditions. While not actively poisonous, it is incapable of supporting life. 1000 cub. ft. of nitrogen $= 5·606 \times 14 = 78·48$ lbs.

Carbon.—Symbol, C; atomic weight, 12. Carbon is not a gas, and it is never found free in a gaseous form like hydrogen or oxygen. Charcoal, coke, graphite, and the diamond are all forms of carbon, the diamond being the purest. This element is often present in compound gases, and, from their properties, gaseous carbon is assumed to have no colour, taste, or smell, to be inflammable, but a non-supporter of combustion.

Sulphur.—Symbol, S; atomic weight, 32. Sulphur is also a solid element at ordinary temperatures; at higher temperatures it becomes a liquid with a clear amber colour, which on continuous heating becomes darker, and at a temperature of 840° F. it becomes a dense red vapour which is combustible, and a non-supporter of combustion and life, without smell itself, but with a strong pungent smell if allowed to combine with oxygen or with hydrogen. Its combination with oxygen constitutes its chief claim to importance as regards mine gases.

The only naturally occurring mechanical mixture we have to deal

with in mine gases is air, which cannot be correctly expressed by any formula.

Air is composed approximately of 4 volumes of nitrogen and 1 volume of oxygen, 14·43 being its relative weight as compared with hydrogen. In addition to these two gases air also contains several other constituents, such as carbonic acid gas or choke-damp, water vapour, and *argon*, a constituent recently discovered by Lord Rayleigh. The proportion of carbonic acid gas (CO_2) in the air is about $\frac{1}{2500}$th part of the whole volume, or varies from 2 to 10 vols. in 10,000 vols. of air.

The average composition of normal air is :—

	Volumes per 1000.
Nitrogen,	779·0600
Oxygen,	206·5940
Aqueous vapour,	14·0000
Carbon dioxide,	·3360
Ammonia,	·0080
Ozone,	·0015
Nitric acid,	·0005
	1000·0000

The average amount of CO_2 present in the air is ·04 per cent.; in ordinary mines, 0·78 per cent.; and in badly-ventilated mines, 2·73 per cent. As a continual supply of CO_2 is being given off from many sources, it is necessary to have some provision made to keep down the amount present in the atmosphere. Nature itself makes the necessary provision. Plants inhale the CO_2 present with the absorption of the carbon and some of the oxygen, the compound retained being assimilated, and helping to build up the plant tissue.

Moisture in the Air.—There is always a certain amount of water vapour present in the atmosphere, but the quantity is subject to great variation. The barometer gives indications as to the condition of the atmosphere in this respect. The amount of vapour or moisture which the air can take up depends on its temperature: the higher the temperature the more water can be held in suspension. There is, for any given temperature and pressure, a maximum amount of moisture which a given volume of air is capable of taking up, and at which it is 'saturated.' The following quantities of water correspond to 'saturation' for the temperatures given :—

Degrees F.	Weight of water in lbs.
1000 cubic feet of air at 32° contain	0·308 lb.
,, ,, 50° ,,	0·567 ,,
,, ,, 68 ,,	1·066 ,,
,, ,, 86 ,,	1·873 ,,

When air saturated with vapour is cooled, the moisture is condensed and falls in the form of rain or dew.

The relative weight of water vapour to air is as 9 to 14¾. Water vapour is therefore much lighter than air, and a column of moist air

is much lighter than a column of dry air of the same height. When the barometer falls it indicates a decrease in local pressure, because the air is moist and there is a probability of rain, whilst when the barometer is high it indicates that the air is dry and that dry weather will occur.*

The amount of vapour in the air can be ascertained from tables published for the purpose (the physical tables edited by Prof. Guyot, of Washington, are the best), giving the average saturation for different temperatures. But probably the best means of dealing with this is to find by means of calcium chloride tubes carefully weighed before and after a known volume of air has been passed through them; the amount of moisture present in the intake and also in the return can be thus ascertained, and the difference between these two quantities will be the amount of moisture absorbed from the underground workings. The amount of moisture in the air can also be ascertained by finding the dew point, for the intake and return currents, by means of a hygrometer.

Example.—A ventilating current of air of 150,000 cub. ft. per minute saturated with vapour passes down the down-cast shaft at a temperature of 32° F. When it leaves the up-cast its temperature is 75° F. and it is still saturated with vapour. Find how much water-vapour this quantity of air has absorbed from the underground workings.

By the formula $Q_2 = \frac{T+459}{t+459} \times Q_1$, the quantity or volume of air in the up-cast

$$\text{shaft} = \frac{75+459}{32+459} \times 150000 \qquad \left\{ \begin{array}{l} Q_1 = \text{quantity of air entering originally.} \\ Q_2 = \text{quantity leaving the shaft.} \\ T = \text{temperature of air in the up-cast.} \\ t = \quad ,, \qquad ,, \qquad \text{down-cast.} \end{array} \right.$$

$$= 1{\cdot}0875 \times 150000$$

$$= 163125 \text{ cub. ft. per minute}$$

From the tables already referred to it is found that

1 cub. ft. of air at 32° F. contains 2·20 grains of vapour
and 1 ,, ,, 75° F. ,, 9·41 ,, ,,

$$\therefore \text{ vapour in down-cast volume} = \frac{150000 \times 2{\cdot}20}{7000} = 47{\cdot}10 \text{ lbs. } \{7000 \text{ grains} = 1 \text{ lb.}\}$$

$$,, \quad \text{up-cast} \quad ,, \quad = \frac{163125 \times 9{\cdot}41}{7000} = 219{\cdot}05 \ ,,$$

∴ vapour absorbed from mine workings = 219·05 - 47·10 = 161·95 lbs., or 16·195 gallons of water per minute.

The compound gases found in mines are four in number, viz., carbon dioxide (CO_2), carbon monoxide (CO), sulphuretted hydrogen (H_2S), and carburetted hydrogen or methane (CH_4).

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* *Height of the Atmosphere.*—We are quite unable to tell to what height the atmosphere really extends, but we can readily estimate its height, from the observed pressure, if we assume it to have a uniform density.

The average pressure of the air at the sea-level is 14·7 lbs. per sq. in. at a temperature of 32° F. and 29·9 in. of mercury.

14·7 × 144 = 2116·8 lbs. pressure per sq. ft., 1000 cub. ft. of air weighs 80·728 lbs. ∴ 1 cub. ft. $= \frac{80{\cdot}728}{1000} = {\cdot}08072$ lbs., and $\frac{2116{\cdot}8}{{\cdot}08072} = 26211$ ft. = height of air column if the atmosphere were of uniform density throughout.

Black-damp, Choke-damp, or Stythe, is a gas, or, more correctly, a mixture of gases, frequently met with in mines, especially in old workings or badly ventilated parts of the mine. In fact, in mostly all mines it is generally present to a greater or less extent in the return air currents. Until quite recently black-damp was supposed to be composed of pure carbon dioxide or carbonic acid gas (CO_2), but the investigations and analyses of Dr John Haldane have shown that this is not the case. In a series of analyses of samples taken from the underground workings of several collieries it has been shown that the composition of black-damp was very regular, and consisted of 85 to 88 per cent. of nitrogen and 12 to 15 per cent. of carbon dioxide. The following table, taken from Dr Haldane's published researches, will show more clearly the composition of this gas :—

TABLE SHOWING THE COMPOSITION AND SPECIFIC GRAVITY OF BLACK-DAMP.

Component Gases.		I.	II.	III.	IV.	V.	VI.
Composition of the sample	Oxygen, . .	1·45	0·72	10·07	9·60	13·66	13·60
	Nitrogen, .	82·56	80·78	82·30	83·08	78·97	80·68
	Carbon dioxide,	10·64	11·03	7·63	7·32	4·49	4·82
	Marsh gas, .	5·35	7·47	0·00	0·00	2·88	0·90
		100·00	100·00	100·00	100·00	100·00	100·00
Calculated specific gravity of sample,		1·0106	1·0030	1·0274	1·0258	1·0029	1·0129
Calculated composition of the pure black-damp	Nitrogen, .	87·87	87·65	85·30	86·48	85·86	85·90
	Carbon dioxide,	12·13	12·35	14·70	13·52	14·14	14·10
		100·00	100·00	100·00	100·00	100·00	100·00
Calculated specific gravity of the pure black-damp		1·0390	1·0403	1·0534	1·0468	1·0502	1·0500

It will be seen that the specific gravity of black-damp is very much lower than was hitherto supposed to be the case. The specific gravity was usually taken at 1·52 (air = 1), which is the density of pure carbon dioxide, whereas it is now shown to vary from 1·0390 to 1·0534, or only about 4 or 5 per cent. higher than air. Black-damp as actually met with in mines may, moreover, be sometimes lighter than air, in consequence of admixture with fire-damp.

Black-damp is produced by the decomposition or combustion of carbonaceous matter in a free supply of oxygen, and is formed in mines by the decay of organic matter, by the exhalations of men and animals, the gaseous products resulting from burning lamps and blasting operations—in fact, wherever combustion is going on. In some mines it is abundant, and is given off naturally from the strata like fire-damp.

According to Dr Haldane, the two principal theories which may be advanced to account for the presence of this gas in mines are—(*a*) that the formation of black-damp is due to the oxidation of coal or associated strata; (*b*) that it is evolved from the coal or associated strata. He regards the first—the oxidation of coal—as being the principal and most likely cause of the origin of black-damp in coal mines. It is well known that many kinds of coal when exposed to air undergo a slow process of oxidation; in fact, the oxidation may proceed so rapidly as to give rise to a considerable increase in the temperature and finally to spontaneous combustion, causing what are known as gob fires. He maintains that black-damp is nothing but the residual gas left by this slow process of oxidation of the coal, at ordinary temperatures. Black-damp does not issue at high pressure from freshly-cut coal in the same way as fire-damp; and coal which has had ample time to drain off all its other gases, may still continue for months and years to produce black-damp. On the other hand, the gas is frequently met with in metalliferous mines and in fire-clay, limestone, ironstone, and other measures where no coal seams are present.

Black-damp has neither colour, taste, nor smell, except when present in large quantities, when a slightly acid taste is experienced. This would, however, be no guide to its detection in mines. It does not burn, nor does it support combustion or respiration. It can be easily detected, as when lights are lowered into it they become black and smoky, or are extinguished. Another test is to pass a quantity of the suspected air through lime-water, when, if the gas is present, the water will turn milky.

The proportion of black-damp in air required to extinguish the flame of a lamp varies according to the percentage of oxygen present. Professor Clowes states that it requires the presence of 15 per cent. of black-damp in the air to extinguish a flame, while Dr Haldane gives the percentage required for the extinction of a flame at 15 to 19 per cent., according to the kind of light used. The following table gives the result of some of Dr Haldane's experiments on extinctive percentages of black-damp:—

Component Gases.	Upright Candle Extinguished.	Oil Lamp (bonneted clanny) Extinguished.	Hydrogen Flame Extinguished.
Oxygen,	17·64	16·43	11·41
Nitrogen,	80·15	79·25	79·53
Carbon dioxide,	2·21	2·49	5·37
Marsh gas,	0·00	1·83	3·69
	100·00	100·00	100·00
Percentage of black-damp,	15·60	19·56	41·72

Black-damp is always difficult to deal with, especially in the case of dip workings, as it may settle near the floor, and a current of air passing over it may fail to remove it. Its presence should always be suspected in such workings (especially if old and unventilated), and at the bottom of wells and sumps. It may be removed from sumps by lowering a bucket containing quicklime, which absorbs large quantities of CO_2, or by letting a quantity of water fall down the shaft, thus producing a strong current of air to displace it. The latter method can only be adopted when there is plenty of pumping power available to raise the water to the surface again.

Carbonic Oxide, Carbon Monoxide, or White-damp.—Composition, CO; atomic volume, 14. This gas is also colourless and tasteless; but sometimes possesses a sweet and delicate odour, especially when present in large quantities. It is a combustible gas, burning in air with a characteristic blue flame and forming carbon dioxide, and is a non-supporter of combustion. Fortunately this gas is found only in exceptional circumstances, such as underground fires, etc. Carbonic oxide is formed by the combustion of carbon with a deficiency of oxygen, or, briefly, is the result of incomplete combustion. Small quantities of this gas are also given off on the explosion of gunpowder. It is usually a constituent of after-damp, and it is said to be given off naturally in some metalliferous mines, and has been found in tunnels during driving operations. Although this gas is inflammable, it cannot be detected by the flame of a lamp until there is about 12 per cent. present in air, whereas much smaller quantities are fatal to life. Carbon monoxide is an extremely poisonous gas; very small quantities present in the air rapidly give rise to severe headache and giddiness, with palpitation of the heart, and if breathed for any length of time, insensibility and death quickly follow. It has been proved that as a very small percentage of this gas affects small warm-blooded animals more rapidly than man, mice or small birds should be utilised in the detection of this gas, the mouse or bird being carried in a small cage, or inside the gauze of a safety lamp. If on entering the foul atmosphere the animal becomes incapable of motion, it should be regarded as a sign of real danger. It should be remembered that while an atmosphere containing 1 per cent. of CO would be almost immediately fatal if breathed, a very much smaller percentage would be equally fatal if breathed for a sufficient length of time; 0·5 per cent. will saturate the blood as well as 1 per cent., but with the lower percentage it will take a much longer time for saturation to be completed. With about 0·06 per cent. of CO in the air, the blood of a man becomes 30 per cent. saturated after 1½ hours. 0·1 per cent. will give 50 per cent. saturation; with 0·2 per cent. the saturation point is increased to 67 per cent., which would soon bring about unconsciousness and death. With 1 per cent. of CO in the air, saturation of the blood would take place in five or six minutes, and death would rapidly supervene.

The main thing for the student to remember is that the small proportion of CO necessary to produce a fatal result cannot be detected by the flame of a lamp, since a flame cap will not form until there is considerably over 1 per cent. present.

If a person is suffering from carbon monoxide poisoning, pure oxygen should be administered, stimulants given to act on the heart and stomach, and the victim wrapped in warm blankets, and hot water bottles applied. Bringing any one suffering from the effects of this gas suddenly into the fresh air may prove very dangerous, and even may prove fatal. Why this occurs has never been satisfactorily explained, but still it is a fact which has been noted in the case of several colliery accidents.

Sulphuretted Hydrogen.—Composition, H_2S; atomic volume, 17. This is a combustible gas burning with a deep blue flame, producing sulphur ~~dioxide~~ (SO_2) and water; ~~does~~ not ~~support~~ combustion, and has no colour, no taste, but a very strong odour of rotten eggs. Like carbonic oxide, this gas is never found in large quantities in mines. It is produced by the decomposition of pyrites by water. It is an exceedingly poisonous gas, a very small percentage causing sickness and giddiness. It is never a source of great danger in coal mines, as its presence, even in small quantities, is easily detected, owing to its strong smell. Sudden outbursts of this gas have been known to occur, however, in copper and salt mines, causing loss of life.

Carburetted Hydrogen, Methane, or Marsh Gas, is known amongst miners as 'fire-damp,' 'fire,' or 'gas.' Composition, CH_4; atomic volume, 8. Fire-damp is a gas with neither colour, taste, nor smell. It is highly inflammable, and a non-supporter of combustion.

Carburetted hydrogen is found in petroleum, and is given off when the oil is taken out of the earth and the pressure removed. It is also found in marshes (hence the name marsh gas) as the result of the decay of vegetable matter. Vegetable matter is principally composed of carbon, hydrogen, and oxygen, and when it undergoes decomposition in the air, in a free supply of oxygen, the final products formed are carbon dioxide (CO_2) and water (H_2O). When the decomposition process takes place *without* access of oxygen, such as under water, carburetted hydrogen (CH_4), which is a reduction product, is formed. This explains why this gas is held in the coal and given off naturally when the mineral is worked. The association with carburetted hydrogen of free hydrogen, oxygen, nitrogen, and also heavy hydrocarbons, is said to be due to different stages of carbonisation of the vegetable matter, or it may have been produced by the simultaneous decomposition of animal matter.

Blowers.—As the formation of coal must have proceeded under a complete covering of layers of mud, sand, etc., it is evident that the gaseous products accompanying these changes must have collected and then filled, under considerable pressure, not only joints and fissures in

the seam and surrounding rocks, but must have permeated the coal itself. The hissing and crackling noise observed at the face of freshly-worked coal shows that the occluded gases are held under a certain pressure. Where the pressure is great the gas issues from the coal with a hissing noise, like that of steam escaping, a vent of this description being called a 'blower.' These blowers often continue for lengthened periods to give off fire-damp, showing that they must communicate with reservoirs of gas. The pressure has at times been measured, and in some places pressures varying from 460 to 900 lbs. per sq. in. have been recorded. Where gas at these enormous pressures is present, liability to sudden outbursts always exists. Such outbursts are very dangerous, both by their fouling the air currents and dislodging material.

In the black vein seam of the South Wales coal-field large cavities filled with fire-damp are met with, and when the seam is being worked they sometimes burst out unexpectedly, forcing out large masses of coal and dust, and resulting at times in loss of life. Outbursts also occur from the floor of seams in such large volumes and with such suddenness as to foul the air to a dangerous degree and cause serious explosions. When faults are being approached, blowers of fire-damp are not infrequently liberated, being sometimes preceded by an outflow of water. Great care should therefore be taken when working in such circumstances.

In some shallow mines fire-damp is seldom met with, or found only in very small quantities, having probably escaped from such mines through the permeable strata to the surface. In some seams it is, on the other hand, given off in very large quantities, especially from those earliest worked, which usually drain off the gas from the other seams.

Generally, fire-damp is most abundant in seams ~~of considerable~~ depth, being given off naturally from the strata, and also freely from ~~the coal~~ face; it also issues from cracks in the roof and floor, large volumes being given off at times, sometimes heaving up the floor or causing falls of the roof.

Carburetted hydrogen is rarely found in a pure state. It is generally mixed with other gases, principally carbon dioxide and sulphuretted hydrogen.

The small quantities of carbon dioxide found in blowers of fire-damp or in the return air of mines, have apparently little influence on an explosive mixture of carburetted hydrogen and air, according to experiments which have been made with mixtures of fire-damp and air at Bochum and by Kriescher and Winkler at Freiberg, where 5 per cent. of carbon dioxide (CO_2) was found to have no effect. This is probably also the case as regards other gases associated with fire-damp, such as nitrogen, such gases seeming to act in the same way as an excess of air, merely as a diluent, and tending to reduce the temperature of the explosion.

The experiments of Mallard and Le Chatelier, made on behalf of the French Fire-damp Commission, show that mixtures of CH_4 and air begin to be sharply explosive with 7 per cent. of marsh gas (1 vol. of $CH_4 + 12$ vols. of air), the maximum being reached with 10·8 per cent. (1 vol. of $CH_4 + 8·3$ vols. of air), and ceases to be explosive with 14·5 per cent. (1 : 5·9). For the inflammability of a mixture of CH_4 and air they fix the lowest limit with an amount of 5·8 per cent. of marsh gas, and the upper limit from 16 to 17 per cent.

When a mixture of fire-damp and air in certain proportions is brought in contact with a naked flame, combustion may result either in the quiet burning away, or in an explosion of greater or less violence; that is, in the rapid translation of flame through the whole mixture.

In the course of the experiments it was found that whilst 2 vols. of CH_4 require for their complete combustion 4 vols. of oxygen, 2 vols. of hydrogen require 1 vol.; 2 vols. of ethane (C_2H_6) require 6 vols. of oxygen. Taking the average percentage of oxygen in the air as 20·7 vols., a mixture of air and each one of these gases, to contain sufficient oxygen to burn each completely and form, therefore, the most explosive mixture, must contain the following amounts of each :—

	Per cent.
Pure hydrogen (H_2)	29·28 vols.
Marsh gas (CH_4)	9·38 ,,
Ethane (C_2H_6)	5·58 ,,

If, therefore, any of these gases be present with CH_4, the percentage of such mixed gases required to form with air the most explosive mixture (explosive maximum) will be greater or less than 9·38, according as the gas is either hydrogen or one of the other hydrocarbons.

The following table shows the results obtained by the British Royal Commission on Accidents in Mines, of varying percentages of fire-damp on a naked light :—

CH_4 in air.	Effect.
2 per cent. . .	Produces slight elongation of the flame.
$2\frac{1}{2}$,, . .	A distinct elongation.
$4\frac{1}{4}$,, . .	Inflames and burns slowly.
6 ,, . .	,, explodes sharply.
9·38 ,, . .	,, ,, with greatest violence and perfect combustion.
20 ,, . .	The flame just burns feebly.
25 ,, . .	Extinguishes the flame.

Mallard and Le Chatelier submitted the question of the temperature of ignition of carburetted hydrogen to a very thorough and complete investigation, and have found the temperature of ignition for mixtures of CH_4 and air to be 740° C. (1364° Fahr.), and that it is

practically constant whatever be the proportions of the constituents. But it was found that ignition does not take place immediately the gas, or even a portion of it, has been raised to this temperature ; the gas must be exposed *some seconds* to the above temperature before explosion takes place. Professors Wüllner and Lehmann, in their experiments on the same subject, found that mixtures of CH_4 and air in certain proportions were more easily ignited by some kinds of wire, when heated, than by others, copper-wire only causing ignition at the moment of its fusion (about 1100° C.), while platinum wire 0·50 mm. diameter ignited at 1480° C. when the mixture was in the proportion of 1 of CH_4 to 14 of air. They also found that the most easily ignited mixtures are not the most explosive, viz., 1 : 9 and 1 : 10, but those containing gas and air in proportion of 1 : 14 (6·6 per cent. of gas). It was also noted that the higher the velocity at which the inflammable mixture was moving, the higher the temperature of ignition. Wüllner and Lehmann also made experiments with electric sparks, and found that open sparks, as produced by an electric current (using a dynamo) between copper wires 3 mm. diameter, ignited mixtures of CH_4 and air in the proportion of 1 : 9 with a current of 18 amperes and above, whilst with a current of 15 amperes or less, occasional sparks caused ignition. The ignition took place more easily with brass and iron wires than with copper, but the ignition is no doubt influenced by the heating of the wires, so that a current of 8 amperes may be dangerous. In using carbon points it was found much more difficult to ignite the mixtures than with metal wire ; in fact, an arc of 10 amperes could be maintained in the most explosive mixture without danger. In view of the very large number of mines which now use electricity for some purpose or other underground, it is very desirable that further investigations should be made on this subject.

Explosion of Firedamp.—When firedamp explodes with a mixture of air (9·38 per cent. CH_4 in pure air), the following reaction takes place :—

Before explosion.	After explosion.
One volume of $CH_4 + 2O_2 + 8N_2 =$	$CO_2 \quad + 2(H_2O) + \quad 16\ N$
	blackdamp + steam + free nitrogen.

or simply $CH_4 + 4O = CO_2 + 2H_2O$.

It is very seldom, however, that the exact proportions of fire-damp and air necessary for complete combustion are present when an explosion takes place, as in nearly every explosion the deadly after-damp has been found to contain, in addition to CO_2 and free nitrogen, varying percentages of carbon monoxide and free hydrogen. With an excess of CH_4, *i.e.* over 9·38 per cent., the question of resulting products when an explosion takes place is a complicated one. There would be incomplete combustion and certainly CO formed, as well as some free hydrogen. Indeed, the reaction would be somewhat similar

to that by which generator gas is got by incomplete combustion of coal. Even if a quantitative analysis could be made it would be different with different temperatures, etc. For instance, in the explosion at Micklefield Colliery, April 30, 1896, when sixty lives were lost, it was found on investigation that forty-six of these had been victims to carbon monoxide poisoning, the CO being present in the after-damp, and not to the force of the explosion at all. Dr Haldane gives it as his opinion that on an average about 70 per cent. of the lives lost in large colliery explosions are due to carbon monoxide poisoning.

Means of Detection.—Fire-damp is usually detected by means of a 'Davy' lamp, or by one of the other numerous safety lamps now used. To detect this gas the flame should be turned down as low as possible in the lamp, because, if there is a large flame, it is impossible to see the 'blue cap' which forms on the top of the flame if fire-damp is present.

The 'cap' increases in length as the proportion of gas increases, until, when there is 6½ per cent. present, the 'blue cap' fills the gauze of the lamp. With this percentage of gas the mixture would be moderately explosive, the violence of the explosion becoming greater as the percentage of gas in the air increases, until it reaches 9·38 per cent., which is the most explosive point. None but experienced men ought to be allowed to act as firemen and examine for gas in underground workings. In collieries ~~where gas~~ has not been detected for twelve months, the examination of the workings may be made with an open light, but this is a practice which ought not to be allowed, as gas may appear at any time. The practice of allowing a fireman to take down a naked light along with his safety lamp, on the understanding that the naked light has not to be carried beyond a certain point, is a practice that cannot be too strongly condemned, and no manager of a colliery ought to allow such a thing, as many accidents have resulted in the fireman making an improper use of the open lamp and carrying it where it ought not to be.

Fire-damp may accumulate in large quantities in the open waste, where pillars are being taken out, and in rise workings or in holes in the roof, because, being very much lighter than air, it seeks the highest point in the workings. If the seam worked has a soft shale roof and a bed of hard rock, or fakes above, as in fig. 408, the soft shale falls at once, when a 'lift' has been taken off the pillar and the wood first drawn, but the rock does not break for some time afterwards. This causes an open space to be left between the fallen shale and the rock, in which fire-damp accumulates, if present, so that when the rock ultimately falls the space is filled up, and the gas is forced down the edge of the waste into the working places, and, if naked lights are used, may cause an explosion. The only method of dealing with this danger is to work with safety lamps.

If the Longwall system of working is adopted, there may be a space formed between the shale and the rock, which contains gas that escapes into the workings, owing to a subsidence of the rock.

To obviate this, it is common to rip one place up to the rock, the road of course being banked up; this road, which, if possible,

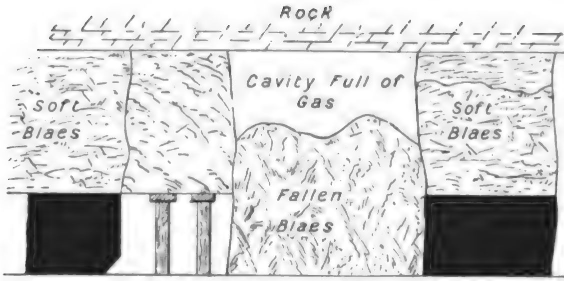


Fig. 408.—Gas-filled cavity.

should be the return airway, has now direct communication with any space that may exist, and consequently acts as a drain for the gas.

In sinking shafts, sudden outbursts of fire-damp have often occurred. When approaching a coal seam the strata are often displaced for a considerable distance from the seam, allowing the gas to issue into the shaft (fig. 409). This may take place more readily after a number of shots have been fired, and therefore great care should be taken in such cases to examine the shaft with a safety lamp before proceeding with the work. The danger may also be guarded against by putting a bore-hole to the coal-head 12 or 18 ft. in advance of the sinking.

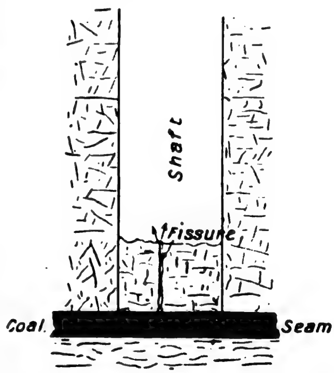


Fig. 409.—Gas vent, or 'blower.'

Another danger lies in pumping water from old workings or disused shafts where gas may be confined, as when the water pressure is lowered to a certain point the pressure of gas may exceed it, and rush out, with dangerous consequences if any naked light be near. A case of this kind occurred at Kinniel Colliery, Bo'ness, a few years ago, whereby two men lost their lives.

Systems of Ventilation.—The different methods of accomplishing ventilation are:—Natural ventilation, waterfalls, furnace or steam jets, reciprocating or displacement machines and fans.

We have seen that air possesses weight; 459 cub. ft. of air at

0° F., and 1 in. barometric pressure, weigh 1·3253 lbs. It is therefore forced along a level road or to the dip more easily than to the rise workings, and this is well known in practice, for with airways of equal area a greater proportion of air will find its way to dip workings.

Since air has weight it possesses the property of inertia, which is the resistance offered by a body to any force tending either to impart motion to it or to influence it when in motion.

*When two shafts are sunk and connected by a passage, and the density (weight) of air in the two shafts is equal, no current of air will circulate, no matter what their respective sizes may be.**

If, however, one column of air has a greater density than the other, then the heavier column will overbalance the lighter one and set a current in circulation.

To cause a current of air to circulate, some means must be adopted to alter the density of one of the air-columns. This may be done in three different ways, viz., by expansion in the up-cast; compression in the down-cast; exhausting the air from the up-cast.

Expansion of the air in the up-cast may be attained by natural ventilation, furnace, or steam jet.

Compression in the down-cast may be procured by means of a waterfall or by compressing fans.

Exhausting the air from the up cast may be carried out by displacement machines or by exhaust fans, the latter being the commonest means employed at mines.

Natural Ventilation.—A mine communicating with the surface by two distinct shafts, having air circulating in them without artificial means, is said to be naturally ventilated. The ventilating current thus set up is caused by a difference in pressure between the two shafts, itself the result of a difference of temperature. Therefore, to have a current of air circulating *naturally*, there must be a difference in depth between the two shafts, to secure a difference in temperature and pressure.

At a short distance below the surface of the earth—50 to 60 ft.—a point is reached where the temperature is constant throughout the year, and as the air acquires nearly the same temperature as the strata, it also is nearly uniform throughout the year.

Descending from this point, the temperature increases on an average 1° F. for every additional 60 ft., and in the case of a mine with two shafts of different depths, it follows that there will be a tendency for a current of air to circulate between them. The air, in travelling round the workings, will also get heat imparted to it by the burning of lamps and the natural heat given off from the strata. Natural ventilation may therefore be due to a difference in depth of the two shafts, due to a difference in surface level, or a difference in depth due to the inclination of the seam.

* *Coal Mining*, by H. W. Hughes, 1899.

Suppose there are two shafts, one 20 fms. in depth and the other 100 fms., the difference in depth being due to difference of surface level. In the winter time, with the temperature at freezing point on the surface, at the bottom of the shallow shaft the temperature would be 52° F., and at the bottom of the deep shaft it would be 59° F.

Now there is a difference in the depth of the two shafts of 80 fms., which would represent a column of air of that height above the surface level at 32° F., opposing a like column of the same height in the deeper shaft at a very much higher temperature, and therefore very much lighter. Let us now examine the pressure exerted in both shafts; the shallow shaft would, of course, be the down-cast in winter, and the deep one the up-cast; the pressure in the down-cast would be—

480 ft. at 32° F.	and 1 cub. ft. of air at	32° F. = ·0811 lb.	∴ 480 × ·0811 =	38·92 lbs.
60 ,, 50° F.	,, 1 ,, ,,	50° F. = ·0780 lb.	∴ 60 × ·0780 =	4·68 ,,
*120 ,, 52° F.	,, 1 ,, ,,	52° F. = ·0778 lb.	∴ 120 × ·0778 =	9·33 ,,
			Total for down-cast =	52·93 ,,

In the up-cast shaft the pressure would be

60 ft. at 32° F.	and 1 cub. ft. of air at	32° F. = ·0780 lb.	∴ 60 × ·0780 =	4·68 lbs.
540 ,, 59° F.	,, 1 ,, ,,	59° F. = ·0767 ,,	∴ 540 × ·0797 =	41·41 ,,
			Total for up-cast =	46·09 ,,

The difference of pressure causing the current to circulate will be 52·83 − 46·09 = 6·84 lbs. in favour of the shallow shaft, which will cause it to be the down-cast in winter; while if we examined the case in the same manner in summer, it would be found that the deep shaft would then have become the down-cast.

The weight of a cubic foot of air at any temperature and height of barometer may be found by the formula $W = \frac{1\cdot3253 \times B}{t + 459}$

Where W = weight of 1 cub. ft. of air at the given temperature.
B = height of barometer in inches.
t = temperature of air in degrees Fahrenheit.
459 = co-efficient of expansion.

1·3253 = a constant (459 cub. ft. of air at 0° F. and 1 in. bar. press. weighs 1·3253 lbs., and 459 cub. ft. of air at 0° F. and B inches of barometric press. = 1·3253 × B).

The more nearly the two shafts approach the same depth or surface level, the more nearly equal will be the pressure in both shafts, causing the current of air to circulate, and hence the greater likelihood of it stopping altogether at times. In mines where ventilation is effected by natural causes, it is no uncommon occurrence for the air

* 120 ft. at 52° F. is only approximately correct, as the temperature would be different at different points in the shaft.

in summer time to be travelling down one shaft in the morning, to come to a standstill in the middle of the day, and circulate in the opposite direction in the evening. In most mines, therefore, natural ventilation is totally inadequate, and should not be depended on where fire-damp or other dangerous gases are given off.

Waterfall Ventilation.—To set a current of air in circulation in a mine by means of a falling column of water is not a method that is very commonly resorted to, nor is it to be recommended, except in cases of emergency, such as the sudden stoppage of a fan. The author has known cases where the plan proved, however, of much service in ridding the workings of a body of choke-damp, but in this case it was only adopted as an auxiliary to the circulation arising from natural ventilation, and had there not been plenty of surplus pumping power to raise the water to the surface again, it could not have been used at all.

This method can only be used economically when there is an adit level connected to the shaft, and which will convey the water from the workings without the employment of machinery.

Furnace Ventilation.—A number of years ago nearly all collieries of any importance were ventilated by means of furnaces, but this system of ventilation has fallen into disrepute, and now few collieries, except in some districts in the north of England, are supplied with air in this manner; and the employment of furnaces for this purpose is within measurable distance of ceasing altogether, as it has little to recommend it.

The action of the furnace is very simple. The current of air, in passing over the fire, is heated, which causes it to expand, and its density being lessened, it can no longer resist the pressure of the denser and heavier column of cold air behind. Hence it is continuously driven forward into the up-cast shaft, and a current thereby established, while the higher the temperature of the heated air, the greater will be the quantity that will pass over the furnace.

The power of a furnace therefore depends on the amount of heat which it can communicate to the current of air; and as the efficiency of a furnace depends on what is known as the height of the motive column, it necessarily follows that the greater the depth of the up-cast shaft, the greater will be the quantity of air passing.*

The motive column is a head of air of the down-cast temperature and of such a height that it will equal the *difference* of the weight between the air in the down-cast and up-cast shafts; or simply, it is the difference in height between two air columns of different temperatures due to their different densities. The motive column can therefore be expressed in terms of its height in lineal feet and inches, in terms of ventilating pressure expressed in lbs. per square foot, and in

* In furnace ventilation the quantity passing varies as the square root of depth of up-cast shaft.

terms of ventilating pressure represented by inches of water gauge, each of these terms being convertible. The following formulæ will show how the motive column may be expressed in each of these terms:—

Let M = height of motive column in feet.
P = pressure in lbs. per sq. ft.
h = motive column in inches of water gauge.
D = depth of up-cast in feet.
d = ,, down-cast in feet.
t_1 = absolute temperature of air in down-cast shaft.*
t_2 = absolute temperature of air in up-cast shaft.
w_2 = weight of 1 cub. ft. of air at 32 Fahr. and 30 ins. barometric pressure (14·7 lbs. per sq. in.).
p = weight of 1 sq. ft. of water 1 in. deep (5·2 lbs.).

Then $M = D \dfrac{t_2 - t_1}{t_2}$ . . . . (1) or $M = \dfrac{P}{w}$ . . . . . (2)

$P = wD \dfrac{t_2 - t_1}{t_2}$ . . . . (3) or $P = M \times w$ . . . . (4)

$h = wD \dfrac{t_2 - t_1}{pt_2}$ . . . . (5) or $h = \dfrac{M \times w}{p}$ . . . . (6)

Example.—If two shafts are each 100 fms. deep, and the temperature of the air in the furnace shaft is at 160° F., while in the other or down-cast shaft it is at 60° F., what would be difference of pressure, and what height of water gauge would this represent?

Taking formula 3 we have the difference of pressure:—

$$P = \cdot 080728 \times 600 \frac{(460 + 160) - (460 + 60)}{460 + 160}$$
$$= 48 \cdot 43 \frac{620 - 520}{620}$$
$$= 48 \cdot 43 \frac{100}{620}$$
$$= 7 \cdot 8 \text{ lbs. per sq. ft.}$$

and the height of water gauge

$$h = \frac{7 \cdot 8}{5 \cdot 2} = 1 \cdot 5 \text{ ins.}$$

These results are only approximately correct, as the weight of a cubic foot of air is taken at 32° Fahr. and 14·7 lbs. pressure per sq. ft.; to get it mathematically correct the weight of a cubic foot of the air would have to be calculated for each shaft and the average taken, but the difference is so small that this may be neglected.

Example.—If the temperature of the down-cast shaft is 50° Fahr., the depth of the up-cast shaft 300 yds., what would the temperature require to be to give a water gauge of 2 ins.?

---

* Zero on the absolute scale may be taken as −492° Fahr. (460 + 32) below the melting-point of ice).

By formula 5 we have

$$h = wD\,\frac{t_2 - t_1}{pt_2},$$

and transposing we have

$$t_2 - t_1 = \frac{hpt_2}{wD}.$$

Substituting now the values we have

$$t_2 - (50 + 460) = \frac{2 \times 5{\cdot}2 \times (460 + t_2)}{0{\cdot}080728 \times 900}$$

$$t_2 - 510 = \frac{10{\cdot}4(460 + t_2)}{72{\cdot}65}$$

Transposing we have

$$(t_2 - 510) \times 72{\cdot}65 = 4784 + 10{\cdot}4t_2$$

and

$$72{\cdot}65t_2 - 37051{\cdot}5 = 4784 + 10{\cdot}4t_2$$

$$72{\cdot}65t_2 - 10{\cdot}4t_2 = 4784 + 37051{\cdot}5$$

$$62{\cdot}25t_2 = 41835{\cdot}5$$

$$\therefore\ t_2 = \frac{41835{\cdot}5}{62{\cdot}25}$$

or

$$= 672^\circ \text{ absolute}$$

$$672 - 460 = 212^\circ \text{ F.}$$

Taking the first example to find the motive column, we have

$$M = 600\,\frac{160 - 60}{160 + 459} = 96{\cdot}9 \text{ ft.}$$

and $p = M \times w = 96{\cdot}6 \times 0{\cdot}08072 = 7{\cdot}82$ lbs. per sq. ft., the same as before.

The water gauge may also be expressed in terms of the motive column, $h = \frac{M \times w}{5{\cdot}2}$. It will be observed that the motive column, as ascertained by the above formula, is at the *temperature of the down-cast shaft*, and at a pressure of 14·7 lbs. per sq. in. Now it is evident that if 'motive column' is to be used as an expression of ventilating power, the motive column of each mine must be reduced to some definite standard. This standard, as given by Mr Atkinson, is the temperature of 32° F. and 14·7 lbs. per sq. in. atmospheric pressure; therefore, to ascertain the motive column *under any conditions of temperature or pressure*, and reduced to above standard, the formula will become

$$S = \left\{ \frac{\left(\frac{1{\cdot}3253 \times B}{459 + t_1}\right) - \left(\frac{1{\cdot}3253 \times B}{459 + t_2}\right)}{{\cdot}080728} \right\} D \quad . \quad . \quad . \quad . \quad (7)$$

where S = standard motive column
B = height of barometer in inches
t_1 = temperature of down-cast shaft
t_2 = ,, up-cast ,,
D = depth of up-cast in feet.

The horse power expended in furnace ventilation may be found by the formulæ

$$\text{H.P.} = \frac{Q \times P}{33000} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (8)$$

$$\text{H.P.} = \frac{Q \times M \times w}{33000} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (9)$$

or

$$\text{H.P.} = QD\left\{\frac{\left(\frac{1 \cdot 3253 \times B_1}{t_1}\right) - \left(\frac{1 \cdot 3253 \times B_2}{t_2}\right)}{33000}\right\} \quad . \quad . \quad . \quad (10)$$

When Q = total quantity of air glowing in cub. ft. per minute.
B_1 and B_2 = height of barometer in up-cast and down-cast shafts respectively.

As already stated, the efficiency of a furnace depends on the height of the motive column; therefore, the greater the depth of the up-cast shaft, the greater will be the total quantity of air passing. The quantity of air given by a furnace also varies as the square root of the depth of up-cast shaft, and also approximately as the square root of the difference of temperature between the up-cast and the down-cast shafts. For instance, if the depth of an up-cast shaft is 25 fms., and suppose it increased in depth to 100 fms., then the increased efficiency would be in the ratio of

$$\begin{array}{rcl} \sqrt{25} & : & \sqrt{100} \\ 5 & : & 10 \\ \text{or } 1 & : & 2 \end{array}$$

Hence an up-cast shaft 100 fms. deep would give twice the quantity of air that one 25 fms. would give, all else being constant.

Take, again, the temperature; suppose the mean temperature of the down-cast is 60° F. and the up-cast 85° F., and assume that the latter temperature has been increased to 160° F., then the quantity of air passing will be in the ratio of

$$\begin{array}{rrcl} & \sqrt{85 - 60} & : & \sqrt{160 - 60} \\ = & \sqrt{25} & : & \sqrt{100} \\ = & 5 & : & 10 \\ & \text{or } 1 & : & 2 \end{array}$$

Hence the quantity of air will be doubled.

Again, the efficiency of a furnace varies directly as the fuel consumption, therefore the horse-power for a given fuel consumption varies inversely as the depth, and may be shown as follows:—

Depth of shaft in ft.	Fuel consumed per horse-power per hour.
250 ft.	96 lbs.
500 ,,	48 ,,
1000 ,,	24 ,,
1500 ,,	16 ,,
2000 ,,	12 ,,
3000 ,,	8 ,,
4000 ,,	6 ,,

This table shows that, before a furnace can become as efficient as an engine working a fan, in regard to the fuel consumed, the shaft would require to be nearly 3000 ft. deep, as an engine for driving a fan, if in good condition, should not, on an average, consume more than 8 to 10 lbs. of coal per horse-power per hour.

The difference in cost and upkeep between a fan and furnace plant, both dealing with the same quantity of air, may be taken as under :—

(1) *Fans, etc.*

	Approximate Price.		
Two double-flued Lancashire boilers 30 ft. × 7½ ft., ...	£750	0	0
Guibal fan 40 ft. × 12 ft. (32 to 35 revs. per minute), with engine complete, . . .	1100	0	0
Buildings, for engine house, fan casing, etc., . .	500	0	0
Sundries, . . .	100	0	0
Total,	£2450	0	0

Cost per annum (365 days).

Interest on £2450 at 5 per cent. . . .	£122	10	0
Depreciation at 5 per cent.,	122	10	0
Repairs, ...	10	0	0
Stores, furnishings, etc.. .	10	0	0
Dross for boilers (3¾ tons per 24 hours at 1s. 6d. per ton), . . .	102	13	1½
Enginemen's wages (two at 4s. per shift), . .	146	0	0
Total,	£513	13	1½

(2) *Cost of Furnace.*

	Approximate Price.		
Estimate for building and material, ...	£150	0	0
Making 'dumb-drift,' etc., ...	100	0	0
Total,	£250	0	0

Cost per annum (365 days).

Interest on £250 at 5 per cent., ...	£12	10	0
Depreciation at 5 per cent.,	12	10	0
Repairs, ...	5	0	0
Triping, 8¾ tons per 24 hours at 3s. per ton, .	479	1	3
Attendants' wages, three men at 3s. 6d. each per day, ...	191	12	6
Total,	£700	13	9

Cost of furnace per annum, ...	£700	13	9
Cost of fan per annum, .	513	13	1½
Difference in favour of fan, ...	187	0	7½

If the item for enginemen's wages (£146) be omitted—and as at most collieries the winding or other enginemen attend the fan along with their other duties—then the difference in favour of the fan per year would be £333, so that the saving per annum would cover the entire cost of a fan installation in about seven years.

The shafts for which the plant in the above estimates was required were 125 fms. deep. In a more shallow mine the fan would compare to even greater advantage.

In the erection of furnaces the greatest care ought to be taken to minimise risk against fire ; the side walls should be 14 in. to 18 in. thick opposite the bars, with plenty of room for a current of air

to circulate outside them, and if the furnace is built near to a seam of coal, it should be provided with a double arch, and space left between for a current of air to circulate through to keep the outside cool; the space between the outside arch and the strata being filled with sand and ashes, which are bad conductors of heat. Fig. 410 shows the construction of a ventilating furnace.

In mines where the return air contains fire-damp in dangerous quantities it must not be allowed to pass over the furnace, as it would ignite and cause an explosion, but should be led into the up-cast shaft by a separate way, termed a 'dumb' drift, at some distance from the furnace drift, or it may enter the shaft on the opposite side from the furnace, higher up the shaft, or by another seam, or from the workings of a lower seam, if the furnace be placed

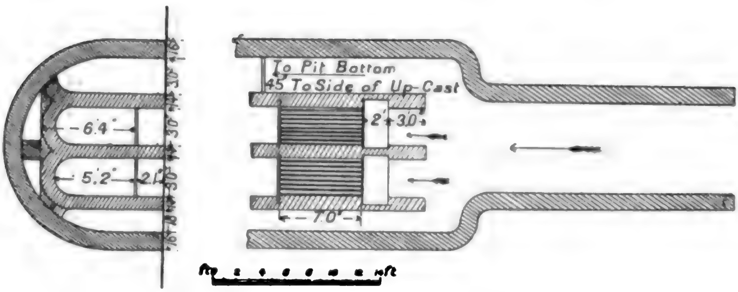


FIG. 410.—Ventilating furnace.

far enough back to prevent the flame of the fire from entering the shaft (figs. 411, 412).

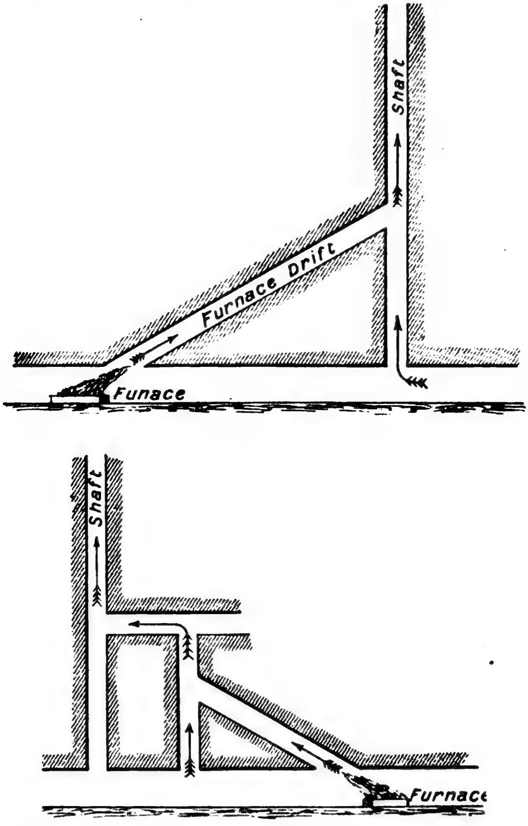
In such cases the furnace must be fed with pure air direct from the down-cast, or from some section of the workings sufficiently pure for the purpose; in either case the efficiency of the furnace would be greatly reduced.

When such is the case, the fire-bars are fitted with doors, so as to force as much air through them as possible, and in this way the air is raised to a higher temperature; if only a certain quantity of air is required to assist combustion, the space below the fire-bars may be fitted with doors, and the air brought from the down-cast in pipes, the supply being regulated so as only to let as much past as is necessary to secure complete combustion. If the fire burns feebly from the presence of black-damp (CO_2), a similar arrangement is necessary.

In cases where iron tubbing is used for lining the shaft, it ought to be protected by an inside lining of fire-brick from the destructive fumes and heated gases given off from the furnace. As fire-brick is a bad conductor of heat, less heat, and therefore less energy, will be

lost than if the air were in direct contact with the iron tubbing during its passage up the shaft.

Speed of Air in Up-cast.—The speed at which the air travels in the up-cast varies very much according to size of the shaft and the extent of the underground workings, and may range from 20 to 50 ft. per



FIGS. 411, 412.—Diagram of furnace ventilation.

second, according to the quantity of air passing, and the area of the shaft.

The use of furnaces underground for ventilation is objectionable and not to be recommended, especially in mines in which fire-damp is given off, the only thing that can be said in their favour being that they ~~are very~~ much cheaper as regards first cost than fans. They have, however, many disadvantages, among which may be noted:—

The possibility of an explosion from a sudden outburst of gas.
Shaft repairs can only be carried out with great inconvenience, if the up-cast is used as a winding pit.
The expense incurred for fuel and attendance is heavy compared with that required for fans.
Where iron tubbing, rope, or iron guides are used in shafts, great damage is done to them by the gases given off from the furnaces, and also by the expansion and contraction due to the variations of temperature.
The circulation of air by means of furnaces may become very unequal, depending often on the management of the man in charge.

Mechanical Ventilation.—Mechanical means of ventilation may be divided into two classes, viz. :—

Statical ventilators = Displacement machines or air pumps.
Dynamical ventilators = Exhausting or compressing fans.

Statical ventilators are now so little used for mine ventilation that they need not be described here.

Fans.—The principle on which all centrifugal fans act is known as centrifugal motion. In the case of a fan the centrifugal force drives the air from the centre, the particles being revolved in a circle by the fan until they are finally thrown off at a tangent, thus creating a partial vacuum at the centre, which causes a further supply of air to enter, and thus establishes a continuous current. In general, such ventilators are used to exhaust the air, and are usually placed at the top of the up-cast shaft, but they may also be used as forcing or compressing machines, in which case they would require to be placed at the top of the down-cast.

Of centrifugal ventilators there are now a large number in use, but those most in favour are the 'Guibal,' 'Waddle,' 'Schiele,' and 'Cappell' types, the others being all, more or less, modifications of one or other of these.

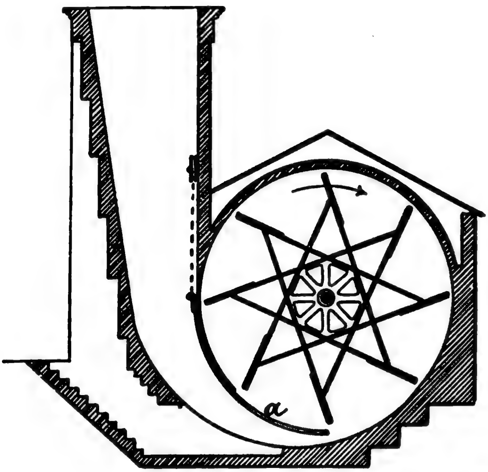
Guibal Fan.—This fan is usually constructed of large diameter, 20 to 45 ft., the width being $\frac{1}{4}$ to $\frac{1}{2}$ of the diameter, while the central opening is about $\frac{1}{3}$ of the diameter.

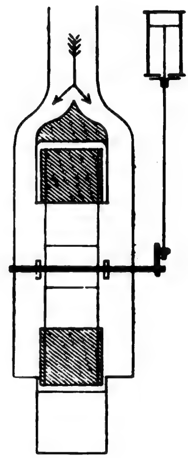
The Guibal fan works in a brick casing, exhausting the air into an évasée chimney, and is now usually fitted with a movable shutter *a*, to regulate the opening for the flow of air.

The fan itself is made of eight or ten straight blades, fitted on to a central casting (figs. 413, 414); in large fans the blades are made of wrought iron, while for small fans, 10 ft. or so in diameter, they may be of wood. The chief objection to this fan is its large size and consequent heavy cost, and also the large outlay for fan chamber and foundations. The great width, moreover, causes a very heavy weight to be distributed over a long length of driving shaft, which can only be supported at each end, and consequently requires very heavy shafting, while instances have occurred where the shafting has suddenly snapped, through the weight. The Guibal fan has, however, from recent experiments, proved to be a very efficient ventilating machine

under widely varying conditions. The speed of the fan varies with the size, a fan 30 ft. diameter being usually driven at from fifty to sixty revolutions per minute, and one of 40 to 45 ft. diameter at forty to fifty revolutions per minute.

Waddle Fan.—This type of fan is of the large open running description, and is the best of this class of ventilator. It requires no fixed casing or chimney, but delivers the air all round its circumference direct into the atmosphere, and therefore its width is reduced at the periphery, which causes it to be very narrow, in proportion to its diameter, compared with the Guibal fan. Its diameter varies from 20 to 50 ft., the blades *a b* being 12 in. to 18 in. wide at the circumference, and $2\frac{1}{2}$ to 4 ft. wide at the centre, and fixed into the



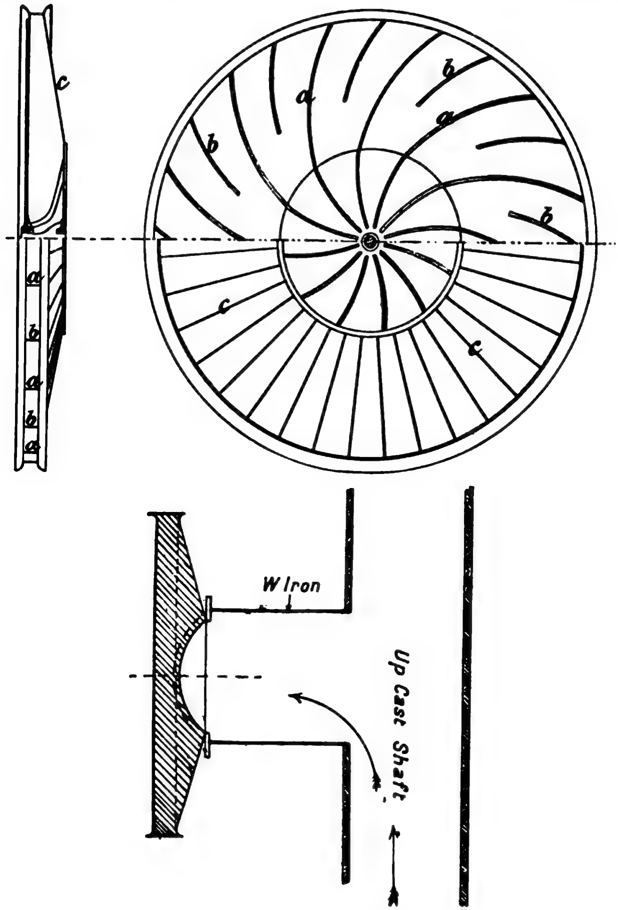


Figs. 413, 414.—Guibal fan.

casing *c c*, both casing and blades revolving in one piece (figs. 415, 416), which tends to reduce the vibration, which is so noticeable a feature in the Guibal fan. All the Waddle fans are now being made with a diverging outlet, *i.e.* the two rims projecting beyond the blades are inclined outwards. This tends to reduce the velocity of the air as it leaves the circumference, and also requires less power to drive it. Its speed is forty to sixty revolutions per minute. Some of the advantages claimed for this fan are :—

Being direct driven, it causes little noise and vibration.
As it requires no enclosed casing or chimney, the first cost of masonry for foundations, etc., is small.
The small cost for repairs per annum.
A high percentage of the power employed is rendered effective.
The vibration and wear and tear are less than in the Guibal fan.

Schiele Fan.—This fan is somewhat like the Guibal in its construction. It has the same expanding chimney, but instead of the blades being straight, they are curved from the centre, and the casing is open, while the width of the blades is not the same throughout,



FIGS. 415, 416.—Waddle fan.

but greatest in the middle, decreasing towards the centre and the tips (figs. 417, 418). The air is taken in at both sides of the fan, which may be driven either direct or by belting, the latter method being the most common. In size the fan varies from 5 to 15 ft. in diameter, with a width at the circumference of 1 to 3 ft., and at the

centre of 2 to 4 ft. The speed varies from 120 to 400 or 500 revolutions per minute, according to size.

Its chief advantages are :—

(1) It is moderate in size, and occupies little space at the surface.
(2) Freedom from vibration and small cost for upkeep.
(3) The shaft bearings, not being in the return airway, are easier seen and more readily available for repairs.

Cappell Fan.—Like the Schiele, this fan is of the small quick-running type. It is constructed of two concentric cylinders, an outer and inner one, each having six curved blades or vanes, the convex sides of which are in the direction of rotation (figs. 419, 420).

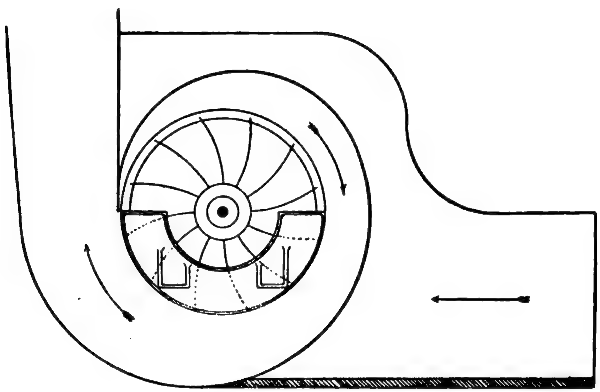


FIG. 417.—Schiele fan.

The cylinders (1) contain port-holes (2) between the two sets of blades, the air passing through these port-holes between the inner and outer chambers; the air is taken in at the centre and thrown off the inner blades into the outer chamber by centrifugal force at a high velocity. It then strikes against the outer blades, and gives back the greater part of the impulse received from the inner blades, which reduces its velocity when discharged. Velocity imparted to the outgoing air always, it should be noted, diminishes the efficiency of the fan.*

In size this fan varies from 8 to 15 ft. in diameter, with a width of 7 to 11½ ft., and it is worked at speeds of 180 to 300 revolutions per minute. The advantages claimed for this fan by the makers are :—

(1) It can do a large amount of useful work in proportion to its size.
(2) Smallness of fan reduces the capital outlay.
(3) It can withdraw large quantities of air.

The latest type of this fan is shown in figs. 421, 422.

* *Ore and Stone Mining*, Sir C. Le Neve Foster, sixth edition, p. 532.

Walker Fan.—This fan (fig. 423) is constructed somewhat like the Guibal, but it also combines other types. It may be termed a medium-sized fan, as it is generally from 20 to 25 ft. in diameter. It is built exclusively of iron and steel. On the main shaft are fitted two strong cast-iron bosses, which extend lengthwise on each side towards the journals, thus distributing the weight of the fan over a considerable portion of the shaft. Between the bosses are placed two

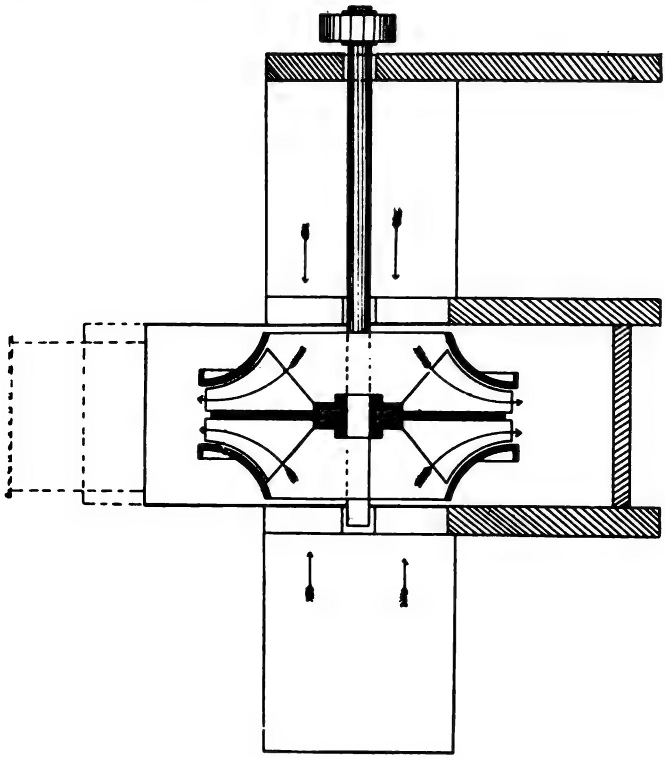
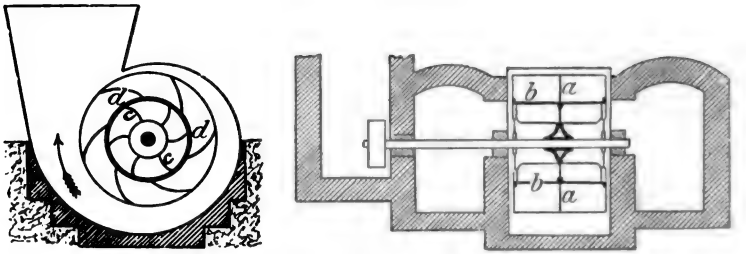


FIG. 418.—Schiele fan.

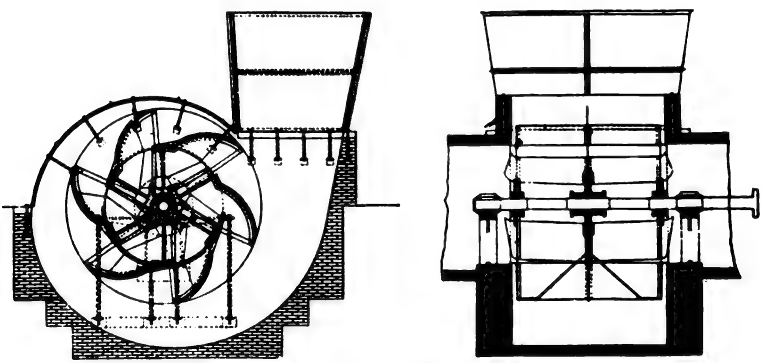
discs of steel, of uniform thickness, bored in the centre to fit the fan shaft. Between these two discs, and gripped tightly by them, are fixed the iron arms of the fan in pairs. These arms extend from near the axis of the fan to its periphery, being supported half-way by the discs. In the small spaces between the discs which are not occupied by the fan arms, there are inserted annular plates. The whole portion outside the boss is then securely riveted together. Angle irons are riveted to the fan arms where they extend beyond

the discs, and to these, eight in number, are firmly secured the cross-section of the arm and vane in the form of a letter T, the top of the T representing the vane and the surface pressing against the air. The vanes, which spring tangentially from a small circle concentric with the fan shaft, are curved longitudinally to the arc of a circle of



FIGS. 419, 420.—Cappell fan.

a certain radius, and are cut away from the edge of the inlet to the fan shaft to minimise central resistance. It is very necessary to minimise the slipping of the air between the sides of the vanes and the walls of the fan chamber. The vanes cannot be brought too close to the walls, as, in the event of any side movement, they might



FIGS. 421, 422.—Latest type of Cappell fan.

catch and be injured. This clearance space is, therefore, filled up by attaching strips of pliable hoop iron to the sides of the vanes.

The fan is also fitted with the Walker anti-vibrating shaped shutter. In the ordinary Guibal fan, as each blade or vane passes the lower edge of the shutter, a pulsatory action of vibration takes place. This vibration is caused by the too abrupt cessation of the delivery of the air from the fan vanes or blades as they pass the opening to the chimney, and for this the shape of the regulating

shutter is responsible. The upper part of this opening, formed by the shutter as hitherto constructed, has a line parallel to the tips of fan vanes, and as the fan revolves these lines become identical; the delivery of the air is, as a consequence, abruptly interrupted. While discharging the air, the pressure is against the front of the vane, but immediately the latter enters the fan casing the load upon it is suddenly removed, and the pressure, owing to the vacuum within the casing, is instantaneously reversed, and causes an upward rebound of the previously depressed blade, with the result that a dangerous degree of vibration is set up. The Walker anti-vibrating shutter, as attached to the fan, removes this evil by effecting a perfectly gradual change in the pressure on the vanes, and so governs the discharge of the air as to cause it to pass, without objectionable eddying, in a continuous stream from the fan vanes into the chimney, instead of intermittently, and without the pulsatory action described.

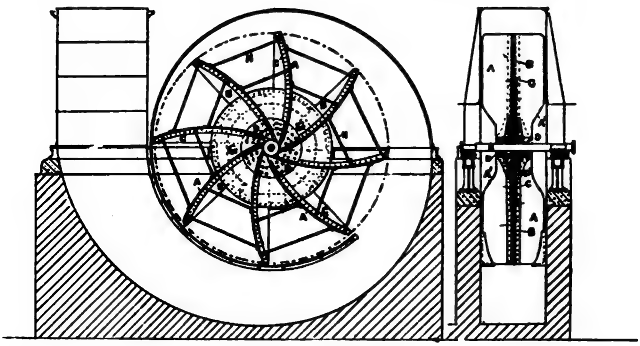


FIG. 423.—Walker fan.

The shutter is constructed in sections, any of which can be removed for the purpose of adapting the area of the opening to varying duties of the fan. The fan is usually driven by rope gearing.

Selection of a Fan.—It is desirable, when erecting a fan to do a given amount of work, to select it so as to suit the varying conditions under which it must act. The height and condition of the underground roads are very important factors in determining the type of fan to be selected, also the amount of room at disposal for surface arrangements; but the principal considerations are: (1) the useful work or effect given out by any fan or engine; (2) the first cost of fan and engine, and of foundations, engine house, fan drift, etc.; (3) the space required at the surface for fan and engine houses, etc.; (4) the relative economy of fuel and stores consumed by the different types of fans; (5) the cost for repairs and freedom from stoppages and breakdowns.

In coming to any decision, especially on the basis of a maker's

estimate, it is nearly always safe to make a deduction of 25 to 30 per cent., taking the amended estimate of the useful work that will be performed, for afterwards, in practical working, it may be found that the maker of the fan has overestimated the merits of his machine by that amount, and it is always better in any case to have a good deal of surplus power in case of an emergency.

Medium-sized and small fans with quick-running engines are now being largely used, and have a good deal to recommend them, as they are economical and safe in working and take up little space. One objectionable feature, however, is the noise which they make while working, which is very disagreeable, especially at collieries situated near to towns. It should also be remembered that small fans cannot produce volumes of air equal to those produced by large-sized fans, as the orifice of discharge, which determines the resistance of the fan to the passage of the air, must necessarily be less for a small than for a large fan. Where large volumes of air are required with a low water-gauge, a fan of large volume will, therefore, be most suitable.

It is desirable in most cases, and in fiery mines almost indispensable, to have a duplicate set of engines, either of which can be immediately attached to the ~~fan in~~ case of a breakdown. Sometimes a duplicate fan is set up, both fans being connected to the fan-drift, so that either can be worked separately for a week or a fortnight at a time. Examples of these arrangements can be seen at Earnock Colliery, Hamilton, and Auchinraith Colliery, Blantyre; at the former there is a duplicate set of engines, while at the latter there is a duplicate fan.

Dimensions of Fan for a Given Quantity of Air.—This is not a matter that is very easily determined, and in erecting a fan it is always as well to get practical details of the work performed by other fans at work in the same neighbourhood, *i.e.* under much the same conditions. Mr A. L. Stevenson says the actual size is to some extent a matter of practical experience, and he recommends the following dimensions for varying volumes of air:—

Under 50,000	cub. ft.	per minute,	30 ft.	diameter.
50 to 100,000	,,	,,	35	,,
100 to 150,000	,,	,,	40	,,

Compressing Fans.—According to the Coal Mines Regulation Act, fans require to be placed some distance back from the mouth of the shaft, and the top of the shaft ought to be lightly covered over to give way easily in the event of an explosion occurring. Fans, as already noted, may be either compressing or exhausting. The former require less power ~~to drive~~ them than the latter. The exhaust fan rarefies the air in the interior of the mine and accelerates the escape of gas from the waste workings. The compressing fan, on the other hand, renders the air in the mine denser and lessens the escape of gas. A falling barometer would intensify the evil of using an

exhaust fan by rarefying the air to a still greater extent. At the same time, it is safer to employ a fan which promotes the escape of gas from the workings, provided there is an ample supply of air available with which to dilute it.

In the case of a colliery where choke-damp is given off in the workings, it might be advisable to use a compressing fan to prevent the escape of this gas into the working faces.

At the Clyde Collieries, Hamilton, experiments were made with compressing and exhausting fans, the results of which gave an apparent advantage to the compressing type, but the experiments were not carried out fully enough to allow of definite conclusions being arrived at.

There are cases, however, where compressing fans have been used to advantage, and give a larger quantity of air than an exhaust fan under the same conditions. One such case, with which the writer is acquainted, was at a small colliery where there were two shafts, one about 90 fms. deep and the other 50 fms.; and as they were separated by a distance of 800 or 900 yards, and no coal was being drawn from the deepest shaft, it was desired to remove the fan which was on this pit to the other one, to save boiler power. This was done; and when it was fitted up, at the 50 fms. pit, as an exhaust fan, it was found to be totally inadequate to supply the quantity of air required. It was then altered to a compressing fan, making the deep pit the up-cast, with the result that nearly 50 per cent. more air was circulated in the workings. In this case it was a distinct advantage to have the fan forcing instead of exhausting the air; but the superiority of the compressing fan in this instance may have been largely due to the assistance it received from natural ventilation, owing to the deeper shaft having been made the up-cast, and the apparent inferiority of the exhausting fan may have been due to the fact that it had to draw the air from the deep shaft through a long stretch of old workings. If a forcing fan is used, the down cast shaft must be covered over, which renders it inconvenient for winding, pumping, and haulage ropes, etc. Again, in the event of an explosion occurring, the disastrous effects and force of the explosion are nearly always felt most keenly in the down-cast shaft; and if the ventilation was carried out on the compressive principle, the fan would in all probability be wrecked by the force of the explosion, putting a stop to the circulation of air when it was most required. On the other hand, with an exhausting fan placed on the up-cast shaft, an explosion might occur and the fan remain intact, which is a great advantage under such circumstances. Taking everything into account, and the fact that the great majority of collieries are ventilated on the exhaustive principle, it would appear that the exhausting fan is the most suitable for general colliery work.

Equivalent Orifice.—The theory of the equivalent orifice of a mine was first investigated by the French engineer M. Margue, and may be stated thus:—The air passing through the workings of a mine meets with a certain amount of resist-

ance. If we imagine that all the air is entering, not through the mine workings, but through an orifice or aperture in a thin plate, then the equivalent orifice for any mine would be the aperture in that plate, *of an area such that a resistance would be offered to the entrance of the air equal in amount to that caused by the mine workings.*

If we call Q the quantity of air in thousands of cub. ft. per minute, h the water-gauge in inches, and A the area of equivalent orifice in sq. ft., then

$$A=\frac{0\cdot37Q}{\sqrt{h}} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (a)$$

$$Q=\frac{A\sqrt{h}}{0\cdot37} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (b)$$

$$\sqrt{h}=\frac{0\cdot37Q}{A}, \text{ or } h=0\cdot1369\left(\frac{Q}{A}\right)^2 \quad . \quad . \quad . \quad . \quad . \quad (c)$$

or, assuming the normal densities of the air and water to be in the ratio of 1·2 : 1000, and Q as before, = the volume of air in thousands of cubic feet per minute; h = inches of water-gauge, then

$$A=0\cdot403\frac{Q}{\sqrt{h}}.$$

Quantity of Air delivered by a Fan.—The quantity of air delivered by a fan is not easy to ascertain, as no two fans, even of the same make, will give the same quantity, even when tried under the same conditions. A great deal depends on the condition of air-courses, type of fan, etc.

The height of the motive column H in feet of air, produced by a difference of temperature or by mechanical means, is the cause of air currents circulating in mines, and the velocity of such air currents (provided there is no resistance or friction) would be equal to that acquired by a body falling from the same height as that represented by the motive column $v=\sqrt{2gH}$, and therefore the theoretical volume of air passing through any opening (equivalent orifice) of area a will be

$$V=va, \text{ but as } v=\sqrt{2gH} \quad \therefore V=a\sqrt{2gH};$$

$$\text{but } H=\frac{u^2}{g} \quad \therefore V=a\sqrt{2g\frac{u^2}{g}}=a\sqrt{2u^2}$$

The theoretical volume produced per second, allowing for the action of the *vena contracta*, will be

$$V=0\cdot65a\sqrt{2u^2}.$$

Applying this to a fan with a tangential velocity in feet per second = u, and a manometrical efficiency = K,

then $$V=\cdot65a\times\sqrt{2u^2}\times K.$$

or if Q = total quantity of air in cubic feet per minute,
d = diameter of fan in feet,
R = revolutions of fan per minute,
u = tangential speed of fan in feet per minute = $d\times3\cdot1416\times R$,
a = equivalent orifice of mine in square feet,
K = coefficient representing the manometrical efficiency,
0·65 = coefficient of the *vena contracta*,

then $$Q=0\cdot65a\sqrt{2(d\times\pi\times R)^2}\times K=0\cdot65\times1\cdot41\pi, a, d, R, K$$
$$=2\cdot88\ a, d, R, K$$
$$\text{or} \quad d=\frac{Q}{2\cdot88\ a, R, K}.$$

Example.—What would be the total quantity of air, in cubic feet per minute, produced by a Guibal fan 35 ft. diameter and moving at a speed of 40 revolutions per minute; the equivalent orifice of the mine being 30 square feet, K being = ·69

$$Q=2\cdot88\times30\times35\times40\times\cdot69$$
$$=83,462\cdot40 \text{ cubic feet per minute.}$$

Example.—What size of fan, making sixty revolutions per minute, and giving an efficiency of 60 per cent., would be required to pass 60,000 cub. ft. of air per minute, the equivalent orifice of the mine being 20 sq. ft.?

This problem may be worked out in two different methods, but we may proceed by the formula, $a = \dfrac{0 \cdot 37Q}{\sqrt{h}}$.

To find the water-gauge—

$$20 = \frac{0 \cdot 37 \times \dfrac{60000}{1000}}{\sqrt{h}} \quad \therefore 20\sqrt{h} = 22 \cdot 20 \text{ and } h = \left(\frac{22 \cdot 20}{20}\right)^2 = 1 \cdot 23 \text{ ins.}$$

We may find the velocity of the periphery of the fan by formula $h = \cdot 00045\, u^2$. Then

$$1 \cdot 23 = \cdot 00045 u^2, \text{ or } u^2 = \frac{1 \cdot 23}{\cdot 00045} \text{ and } u = 52 \cdot 32 \text{ ft. per second.}$$

As the fan is making sixty revolutions per minute, the diameter, supposing its efficiency to be 100 per cent., would be

$$\frac{v \times 60}{R \times 3 \cdot 1416} = \frac{52 \cdot 32 \times 60}{60 \times 3 \cdot 1416 \times \cdot 69} = 24 \cdot 22 \text{ ft.}$$

But the fan has an efficiency of 60 per cent. only, and allowing ·69 for manometrical efficiency,

$$\therefore \text{ the diameter} = \frac{16 \cdot 6 \times 100}{60 \times \cdot 69} = 40 \text{ ft.},$$

so that to pass a volume of 60,000 cub. ft. per minute, with an equivalent orifice of 20 sq. ft., we would require a fan 40 ft. diameter, making sixty revolutions per minute.

Working it out by the more direct formula given above, we have

$$d = \frac{60000}{2 \cdot 88 \times 20 \times 60 \times \cdot 69} = 25 \text{ ft.}$$

and allowing for 40 per cent. in loss of efficiency

$$d = \frac{25 \times 100}{60} = 41 \cdot 6 \text{ ft.}$$

It should be remembered that these formulæ only give approximate results and are not mathematically correct. A more exact approximation of the *actual* volume of air produced by a fan would be found by the formula

$$Q = 40u\sqrt{\frac{2Ka^2O^2}{a^2 + O^2}}$$

where a = equivalent orifice in sq. ft. and O = orifice of discharge of the fan in sq. ft. or an orifice representing the difficulty of the passage of the air through the ventilator.

The value of O is a variable quantity, depending on the equivalent orifice and the water-gauge, and may be found from the expression $O^2 = \dfrac{h_1 a^2}{h - h_1} = \dfrac{ha^2}{0 \cdot 00045u^2K - h}$; h and h_1 being respectively the theoretical and observed water-gauges.*

* Students who desire a fuller treatment of fan ventilation should consult *Theories and Practice of Centrifugal Machines*, by D. Murgue (E. & F. N. Spon, 1883).

The water-gauge (theoretical) that can be obtained from a perfect fan may be found from the formula $h=\frac{v^2}{g}$, where h = ventilating pressure in height of air column, and v^2 = velocity of periphery of fan in feet per second.

Take the speed of fan periphery at 90 ft. per second, $h=\frac{90^2}{32}=253{\cdot}12$ feet, and with a column of 253·12 ft. of air, say at 70° F. and 30 in bar., the pressure $=\frac{1{\cdot}3253 \times 30}{70 \times 459} \times 253{\cdot}12 = 19{\cdot}02$ lbs. per sq. ft., and the W.G. $=\frac{19{\cdot}02}{5{\cdot}2}$ or 3·65 ins.

The same result may be arrived at by the formula $h=\frac{d \times \text{WG}}{d_1 \times 12}$, where h is the height of motive column as before; W.G. the height of water-gauge in inches; d the density of water = 1000; and d_1 the density of air 1·2.

The effective water-gauge = theoretical W.G. × K (or manometric efficiency) and the value of K for different fans varies for different sizes. The manometrical efficiency is the ratio of the water-gauge observed in practice to the theoretical water-gauge deduced from calculation as shown above, or by the formula $h = \frac{v^2 w}{5{\cdot}2g}$; w being the weight of 1 cub. ft. of air at temperature of shaft.

Relatively, the value of K for Guibal fans is ·69, that for Waddle fans ·60, and for Schiele fans ·50.

In fans of the Guibal type the *actual* water-gauge is often more than the theoretical W.G. (from $\frac{1}{10}$ to $\frac{1}{8}$), but this is not always the case, as it is sometimes less.

The amount of *useful effect* produced by a fan is found by carefully measuring the quantity of air put in circulation and measuring the water-gauge; the H.P. in the air $=\frac{\text{Q} \times \text{W.G.} \times 5{\cdot}2}{33000}$.

While the air measurements are being taken, the speed of the fan engine is carefully noted and *indicator diagrams* taken, from which the mean effect of steam pressure acting in the cylinder is ascertained.

Then the H.P. of engine $=\frac{\text{D}^2 \times {\cdot}7854 \times \text{P} \times \text{L} \times 2 \times \text{R}}{33000}$, where D is the diameter of cylinder of engine in inches; P, the effective pressure of steam acting on piston in lbs. per sq. in.; L, the length of stroke in feet; and R, the revolutions of crank per minute.

Then the useful effect of fan will be as H.P. in air : H.P. in engine.

Example.—If the total quantity of air passing is 160,000 cub. ft. per minute with a W.G. of 2 in., the fan engine being 20 in. diameter with a 3½ ft. stroke, going at forty revolutions per minute, with an effective steam pressure of 40 lbs. per sq. in., what would be the efficiency of the fan?

$$\text{H.P. in air} = \frac{160000 \times 2 \times 5{\cdot}2}{33000} = 50{\cdot}42$$

$$\text{H.P. in engine} = \frac{18^2 \times {\cdot}7854 \times 40 \times 3{\cdot}5 \times 2 \times 40}{33000} = 86{\cdot}36.$$

The efficiency of the fan will therefore be as 50·42 : 86 36 or ·58, which multiplied by 100 = 58 per cent.

The quantity of air produced by a centrifugal ventilator, if the speed remains constant, varies (1) inversely with the resistance of the mine, and becomes zero when the resistance of the mine is infinite or when the inlet from the mine is completely closed; and (2) varies as the orifice of the fan (O) or the orifice of the mine (a) is increased or decreased. The quantity of air produced also varies with the speed of the fan, other conditions remaining constant. The above formula also demonstrates that the quantity of air produced by small fans can never equal that produced by large fans, as the orifice of the ventilator, which measures the resistance of the fan to the passage of the air, must necessarily be less for a small than for a large ventilator, hence better and more efficient results can be got by employing medium-sized fans, 20 to 30 ft. diameter, especially where the seams worked are a fair height and where the airways are of fairly good dimensions and can be kept in good order.

Speed of Fan.—This will altogether depend on the size and weight of the fan, for there is a limit to the angular velocity at which a fan may be driven, and to go beyond which would be dangerous. As a rule the velocity of the periphery may vary from 90 to 110 ft. per second, the latter being a practical limit for medium or large-sized fans.

Advantage of Fans over Furnaces.—(1) They will not ignite gas; (2) are more under command and better suited for repairs; (3) the quantity of air passed is more regular and more easily varied at will; (4) less cost incurred for upkeep and attendance, and smaller consumption of fuel, stores, etc.

Cost of Fans. — The price of fans is a very variable item, but the following are taken from some makers' estimates for fans to deliver 120,000 cub. ft. of air per minute, with a water-gauge of 3 in.*

Schiele Fan, 10 ft. diameter, with engine 20 in. diameter × 24 in. stroke, with driving belt, etc.,	£600
Waddle Fan, 21 ft. diameter, with engine 20 in. diameter × 21 in. stroke, complete,	590
Guibal Fan, 27 ft. diameter, with engine 18 in. diameter × 36 in. stroke, complete,	350
Chandler Fan, 13 ft. diameter, with patent silent running Chandler engine,	700
Walker's Patent Fan, with patent anti-vibration shutter, engine 16 in. diameter × 14 in. stroke,	503
Cappell Fan, 12 ft. × 5 ft. 6 in. open running, with semi-compound engine 16 in. diameter × 36 in. stroke,	360

In addition to the above prices, the brickwork for a Guibal fan would cost on an average £130 to £150, and that for a Cappell fan about £70.

* These prices were quoted in 1896; in 1900 they were 20 to 25 per cent. higher. At the present date (1905) they may be taken about the same price as for 1900.

Auxiliary Underground Fans.—In underground work it frequently happens that long close or narrow drifts require to be driven in the solid coal or in stone-work, such drifts being only in communication with the main ventilating current at one end. When this occurs the drifts are difficult to ventilate—especially if they are long and fire-damp is given off—even with the aid of an efficient mid brattice. Under such circumstances a small auxiliary fan may be employed to assist the ventilation. If the quantity of air required for such work is not large, then a small fan rotated by manual labour may be employed, but if the drift is of considerable length and requires a larger quantity of air to keep it sufficiently ventilated, a fan of larger dimensions, say 2 to 3 ft. diameter, will require to be used, driven either by a compressed air or electric motor. The latter is the preferable method, owing to the ease with which a small motor can be installed at any position desired, and the higher efficiency which can be got from it, compared with a compressed air motor. The fan may be connected to the drift by either a dividing wood or brick brattice, wooden boxing, or large sheet-iron circular tubes, $1\frac{1}{2}$ to 2 ft. diameter.

Method of Driving Fans.—Fans working on the surface may be driven either—(*a*), direct; (*b*) by belting; (*c*) by ropes.

Large fans of the Guibal or Waddle type, which are run at moderate speeds, are usually driven direct; *i.e.* the connecting rod of the engine is connected direct to the fan shaft through a crank or disc, and this, on the whole, gives good results and saves the extra cost of driving pulleys and belts or ropes. Small quick-running fans, of the Cappell or Schiele type, are usually driven by belts or rope gearing, the latter giving the best results, for although ropes are more expensive in first cost, they wear much better and are not so apt to give way as belting. The ropes may be either made of hemp or cotton, the latter being now largely employed.

Guiding and Conducting the Air Current.—In no part of colliery work does so much thought and care require to be expended as in the arrangements for guiding and conducting the different underground air currents. In many collieries large quantities of air can be measured in the intake and return airways near the shafts, but the actual amount that really reaches the working faces—the most important point—is often comparatively small. To ensure the maximum quantity reaching the working faces, the main air current will have to be split into as many currents as are necessary for each individual colliery, and each of those currents carefully conducted to the working faces by means of air-crossings, trap-doors, bratticing, screens, etc.

Air-Crossings.—When two currents, an intake and return, have to cross each other, an air-crossing will require to be constructed, so that they may not intermix. Where the strata are hard and free from open fissures the air-crossings may easily be constructed to pass

over each other (fig. 424), but where the strata are soft and friable, an air-crossing has often to be built of brickwork and wood, or brickwork and iron. Whatever method is adopted, the crossing ought to be thoroughly air-tight.

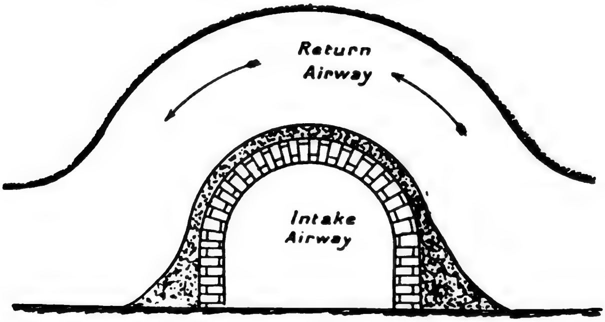


FIG. 424.—Air-crossing.

Trap-Doors.—In all collieries a number of trap-doors are required, either on the main roads or on some of the subsidiary haulage roads. As few doors as possible ought to be used on the main haulage roads, as they are often dangerous in these positions, unless attended by a 'trapper,' and they are also expensive to keep in order and repair, owing to the continual squeeze on the strata, which tends to constantly crush the frames out of position and shape. Fig. 425 shows the method of fixing an ordinary trap-door on underground roads. The door is usually made of wood, 1 to 1¼ in. thick, hinged to frames 6 or 7 in. square. An iron bow is fixed in front for the tubs to strike and to facilitate their passage. All doors should be made so that they will close automatically, which can be easily accomplished if they are properly hung with the right amount of 'lean-to' at the top. Sometimes a weight and pulley are attached to trap-doors to assist in closing them, but unless these are carefully fitted up and often examined they are sure to get out of order, and do more harm than good. Fig. 426 shows a very simple and effective method of automatically closing trap-doors.

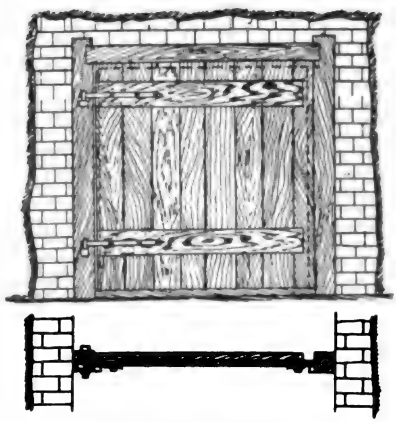


FIG. 425.—Trap-door.

To the door D is fixed a bar of wood A, 6 or 7 in. square, set at such an angle as to clear the approaching tubs at the end unconnected to the door. This end works on a roller pulley B, fixed at the side of the road, and clear of the rails. As the tubs approach they first strike the horizontal bar and gradually push the door open, the sides of the tub rubbing against the bar until they pass through. After they have passed, the door closes of itself, the weight of the attached bar of wood helping to draw it back. This apparatus is easily attached to any door, and will work well on haulage roads,

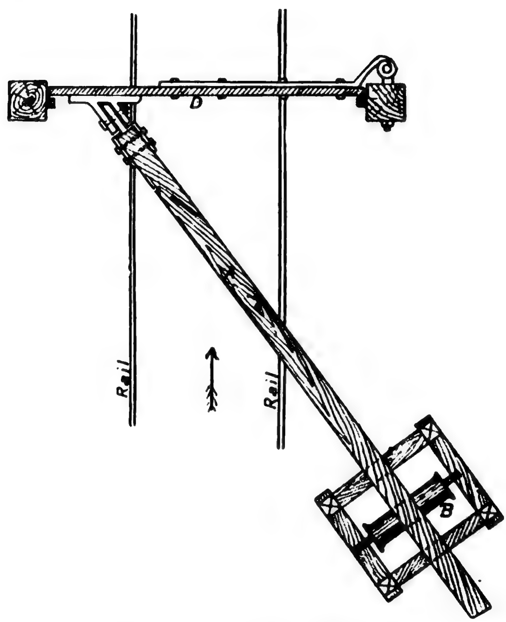


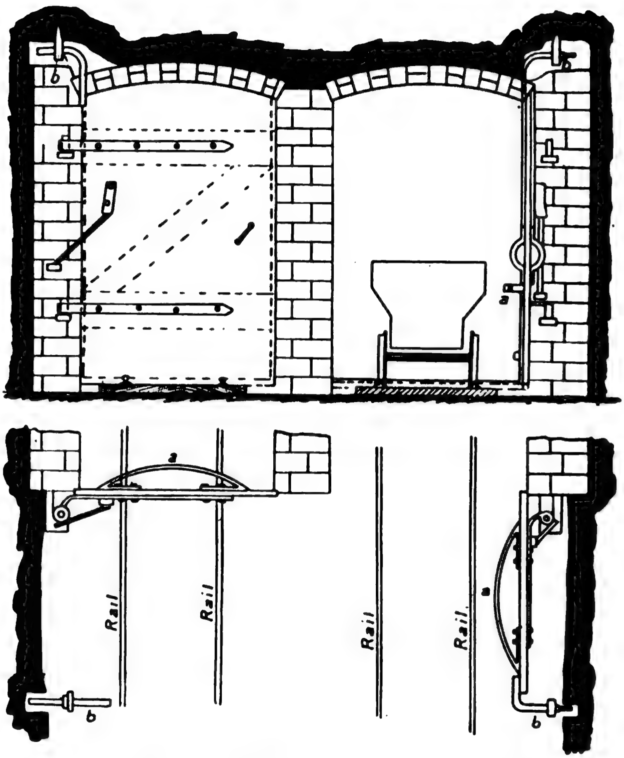
FIG. 426.—Automatic closing door.

especially with slow-speed haulage, without a 'trapper' being required to attend to it.

An automatic door, such as is used for main haulage roads in some districts in Germany, is shown in figs. 427, 428. It is made of wood, covered with felt, and hinged to the brickwork. To facilitate working it is hung slanting. On the inside there is fastened a half-hoop of strap-iron *a*, lower than the top of the tub. Behind the door is fixed a prop, or else a bolt *b*, made of round bar iron and held in a socket fastened to the wall. When the door is opened the bolt is about 4 in. from the edge, and on turning engages with it, and when it closes the bolt hangs down. On the approach of the

tubs the door is opened by the pony-driver, and is caught by the bolt *b*. After the horse has passed the door, the first tub presses against the hoop *a*, and so pushes the door against the wall, in consequence of which the bolt, now liberated, returns to its original position. The door is held open by the tubs as they pass, and after the passage of the last tub closes of itself.

An ingenious self-closing door,* which was first exhibited in

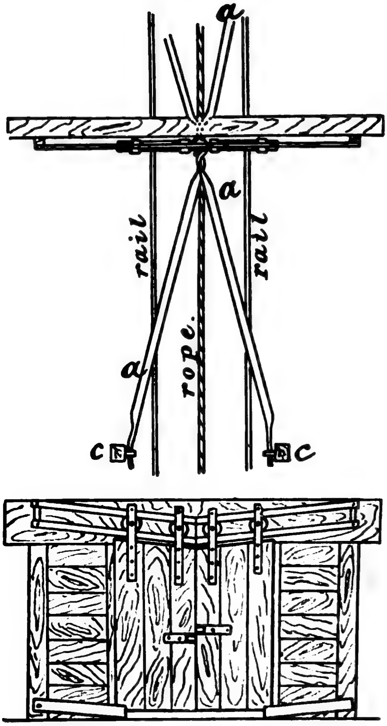


FIGS. 427, 428.—Automatic doors.

working order at the Newcastle Mining Exhibition, is shown in figs. 429, 430. The door is made in two halves, and hung by pulleys which travel on rails, inclined from both sides towards the centre of the road, which allows the two halves to come together by their own weight. About 2 ft. from the bottom of each division, and attached to the edge of each door, is an angle-iron *a a*, 7 or 8 ft. long, the outer end of this angle-iron being attached to an eye-bolt fastened

* *Trans. Min. Inst. Scot.*, vol. ix. p. 123.

to a prop *c c* clear of the rails. As the tubs approach, they first strike the angle-iron, gradually opening the two leaves of the door, which close again as the tub passes out from between the corresponding bars on the opposite side. The door is opened and shut in a similar manner when a tub passes through it in the opposite direction.



FIGS. 429, 430.—Self-closing door.

In the ordinary trap door the tubs often strike very hard, but in this case the doors are opened without shock. On main roads, where the doors serve as the only partitions between the main intake and return airways, there should be at least two doors, each separated by a distance sufficient to accommodate the full train of tubs, and permit the first door to close before the second one opens.

Brattice. — In most underground workings there will be certain places that require air conducted to them by means of 'bratticing'; this is especially the case in pillar and stall workings, or where close or stone drifts are being driven.

When a close or stone drift has to be driven to any considerable length, and where the road has to be divided to form an intake and return airway, the central division is best made of masonry (fig. 431). If the roof is good, and no great crush anticipated, a $4\frac{1}{2}$-in. brick wall may suffice, but it is best to build it $9\frac{1}{2}$ to 14 in. thick. Sometimes only a small brick quadrant is built in roads where traffic is difficult, such as steep inclines or low-roofed ways (fig. 432). A half wall may be built, and a longitudinal plank laid on it, from which boards are laid across to the side of the heading (fig. 433), the seams in the top boards being well filled with good clay, or mortar, and afterwards covered over with brattice cloth.

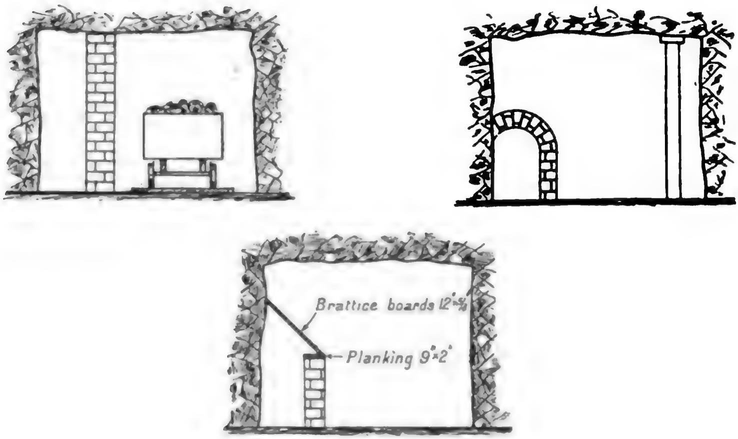
Wood brattice, or a combination of wood with cloth, is largely used for short distances, such as occur in pillar and stall workings.

To fix the brattice cloth, planks 11 in. × $\frac{3}{4}$ in. are nailed along the

top of the props and at the bottom close to the floor. The cloth is fixed tightly to these planks, and also to the props, by means of short, flat-headed nails. The greatest care must be exercised in fitting the joints of the cloth, and in having such joints opposite a post, as well as in the fixing, top and bottom, for unless this sort of brattice is well fitted together, it becomes difficult to convey a current of air for more than 30 or 40 yards.

In places where much fire-damp is given off, it is often difficult to keep the face clear by means of brattice at a distance of 20 yards.

In dividing the road by bratticing, the spaces for intake and return are generally of unequal area; and in such cases it is the general rule to make the narrow way the intake, and the wide tram road the



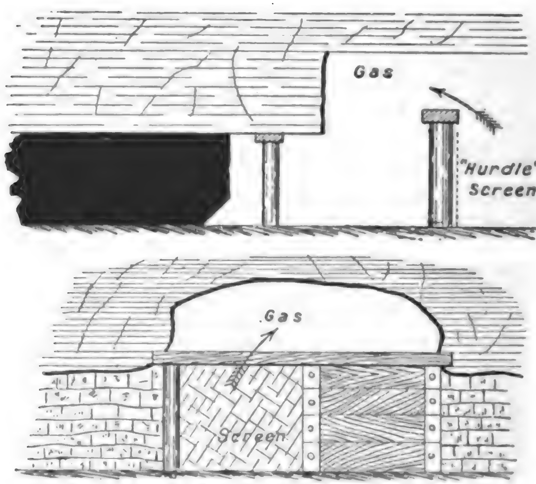
FIGS. 431, 432, 433.—Ventilating close drifts.

return. If the current of air is sluggish, this is undoubtedly the best method, as by making the small area the intake it will increase the velocity, and a supply of air may suffice to keep the face clear, which if brought in by the wider way would have been quite inadequate for such a purpose. If, however, the air can be supplied in large quantities, it can best be brought through the wider way.

In longwall workings, fire-damp often accumulates at the face of the ripping, and in such a case a 'hurdle' screen ought to be set up to clear it out (figs. 434, 435). A hurdle screen is fitted up by fixing a crown or strap across the road, and leaving a space 18 in. or 2 ft. between it and the roof. Two legs or props are set up to the cross-piece, and the screen-cloth firmly nailed to it.

In all branch rods of a longwall working, screens must be fixed

to prevent the air, which is generally required to travel round the faces, from passing away, unless under exceptional circumstances. Fire-damp often accumulates in the open spaces between the packs in longwall, and is often very troublesome. When this occurs, it is cleared out either by conducting a current into such spaces by fixing a screen along the face, or by resorting to 'split buildings.' In this latter method the building or pack wall is not built right along, but at certain intervals a space of 2 to 4 ft. is left, and an airway formed through the waste, so as to clear out any accumulations of gas which may have lodged. Such airways are often difficult to maintain and keep in repair, and new ones will require to be opened at short intervals.



FIGS. 434, 435.

Air Pipes.—Instead of using brattice, wooden boxes or sheet-iron pipes are sometimes used, and are very convenient, but great care must be taken that they are efficient. Where such means are used the volume of air conveyed to the face may be sufficient to clear away the gas at the point of delivery, but at a short distance back from the face, where the velocity of air is much reduced, gas may accumulate in dangerous quantities.

The wooden air-boxes are made from 1 ft. to 2 ft. square, or even larger, and are made of $\frac{3}{4}$ in. boards fitting closely. Sheet-iron pipes from 1 ft. to 2 ft. diameter are also used.

Laws affecting the Air Current.—A current of air, either on the surface or underground, is produced, as has been explained, by differences of pressure or density. A very small difference of pressure or force is required to impart motion to the air, but a current travel-

ling in a circumscribed area meets with a very large amount of resistance, which is termed friction, and of the total force or pressure required to set a ventilating current in motion a very large percentage is taken up in overcoming this friction. Roughly speaking, 90 to 95 per cent. of the actual force is expended in overcoming the resistance met with in the workings, while only 5 to 10 per cent. of the pressure is effective.

If an air current in passing through the underground workings met with no resistance or friction, it would flow with a velocity equal to that attained by a body falling from a given height, and could be expressed by the formula $v = \sqrt{2gH}$, or approximately $= 8\sqrt{H}$, where v = velocity attained in feet per second; H = height in feet fallen through; and g = value of gravity = 32·2. Before air will flow from one point to another, there must be a difference of pressure between the two points, and this difference in pressure can be measured as a head of air or motive column H, which may be substituted for the height, in above formula, through which the falling body has traversed. We have already shown that a head of air or motive column can be expressed in inches of water-gauge $h = \frac{w \times H}{5 \cdot 2}$, w being = weight of 1 cub. ft. of air at the prevailing temperature and pressure. But the value of $w = \frac{1 \cdot 3253 \times B}{460 + t}$ and $H = \frac{v^2}{2g}$; therefore the pressure in inches of water-gauge required to set an air current in motion (with no resistance) will be $h = \frac{\left(\frac{1 \cdot 3253 \times B}{460 + t}\right) \times \frac{v^2}{2g}}{5 \cdot 2}$.

The laws of air friction are usually expressed as follows:—

The pressure required to overcome the resistance due to the friction of air, when the velocity is constant, varies in proportion to the surfaces in contact.

The pressure per unit area required to overcome the resistance due to friction varies inversely, as the sectional area of the airway, or $P \propto \frac{1}{a}$.

The pressure required to overcome the friction varies in proportion as the *square of the velocity.*

Assuming that this latter law is correct (it is approximately correct for all except very high velocities), the force required to overcome friction is expressed by the formula, $F = ksv^2$.

Where F = force of friction measured in lbs.
K = coefficient found by experiment.
S = total rubbing surface of the airway in sq. ft. = total length × the mean perimeter in feet.
v = velocity of flow of air in feet per second.

But $v = \frac{Q}{a}$, Q being = total quantity of air circulating in cub. ft. per second, and a = area of cross section of airway in sq. ft.

$$\therefore F = ks\left(\frac{Q^2}{a^2}\right).$$

If we assume that the resistance is overcome by a head of air, or motive column H, then we can also assume that it is overcome by an urging force $F = Haw$, which is the equivalent of this head.

$\therefore Haw = Ks\left(\frac{Q^2}{a^2}\right)$ (w being = weight of 1 cub. ft. of air at given temp. and bar.), from which we have $H = \frac{k}{w} \times \frac{sQ^2}{a^3}$, or in inches of water-gauge $h = \frac{k}{w^2} \times \frac{sQ^2}{5 \cdot 2(a^3)}$.

The factor $\frac{k}{w}$ is determined by direct experiment in mines.

These are the three principal laws enunciated by the late J. J. Atkinson, and two other laws may be stated, which as a natural consequence follow from them, viz. :—

The power required to overcome the resistance to an air current in the workings of a given mine varies as the *cube* of the velocity, and the resistance measured in lbs. per cub. ft. of air current is equal to the product of the coefficient of friction, the total contact surface and the velocity squared and divided by the sectional area of road, *i.e.* $P = \frac{KSV^2}{a}$.

The rubbing surface spoken of is the whole extent of surface exposed to the ventilating current, *i.e.* the roof, pavement, and two sides of the road, the sum of these four sides in section being called the *perimeter*. The total rubbing surface will therefore be the total length of airway multiplied by the perimeter.

Example.—In the airway 8 ft. wide, 6 ft. high, and 200 fathoms long, what would be the total surface in contact?

Frictional or rubbing surface $= (8 \times 2) + (6 \times 2) \times 1200 = 33{,}600$ sq. ft.

Since the pressure increases in direct proportion to the length, it follows that if we double the length of an airway the pressure will also require to be doubled, to maintain the same quantity of air in motion ; or, if the length be reduced one-half, the pressure required will be halved.

Another point to be noticed is, that the pressure required to overcome the resistance due to friction depends largely on the size and shape of the airways, *i.e.* whether rectangular, square, or circular. In airways of each of these shapes, and having the same area, viz., 78·54 sq. ft., the perimeter of a circle would be 31·416 ft., of a rectangle, 26·12 × 3, 58·36 ft., and of a square, 35·2 ft.

From this it is evident that a circular-shaped airway presents the least area of rubbing surface of the three shapes enumerated, and hence would oppose the least resistance to a current of air passing through it; but, on the other hand, it is the least practicable of the three, and one that is very seldom adopted, except in the case of vertical shafts.

Since the pressure required to overcome the friction varies *inversely* as the area, it follows that in two airways, the areas of which are as 1 : 2, the pressure necessary to overcome the friction of an air current in them will be as 2 : 1, *i.e.* it would take a pressure of 1 lb. in the airway whose area is 1 to do the same work as a pressure of $\frac{1}{2}$ lb. in the airway whose area was 2.

In the case of two airways of equal lengths and perimeters, and consequently of equal rubbing surfaces, the velocity being the same in each case, but having unequal sectional areas, all the conditions upon which friction depends will be equal, therefore the amount of work required to overcome friction will be equal in both cases.

We have assumed that the velocities are equal, therefore the *total pressure* must be the same in both airways; but as the airways have different areas, the pressure per sq. ft. must be different, and will be less in the large than in the small airway.

Example.—In two airways of equal lengths, one 10 ft. × 8 ft. and the other 14 ft. × 4 ft., the areas are as 80 : 56, while the rubbing surfaces are equal, and therefore the total resistance or pressure must be the same in each. Suppose we have a velocity of 350 ft. per minute in each airway, and that the total energy expended in overcoming resistance equals 120,000 ft., the total pressure for each airway will equal $\frac{120,000}{350}$, *i.e.* 342·85 lbs.; the pressure per sq. ft. will, however, be different, for in the 10 ft. × 8 ft. airway it will equal $\frac{342 \cdot 85}{80}$, or 4·28 lbs. per sq. ft., and in the 14 ft. × 4 ft. airway $\frac{342 \cdot 85}{56}$ or 6·12 lbs. per sq. ft., which is in the ratio of 1 to 1·42. From this it will be seen that the smaller airway requires nearly half as much more pressure per sq. ft. than the larger, and that the pressure varies in *inverse* proportion to the sectional area of the airways through which the currents have to pass.

The pressure required to overcome friction is proportional to the square of the velocity. From this it follows that if the velocity be doubled, the pressure required to overcome the resistance due to friction will have to be increased four times; and if we halve the velocity, the pressure required will only be $\frac{1}{4}$.

If we treble the quantity, the velocity must also be trebled, which will increase the original friction nine times.

From the above laws are deduced nearly all the formulæ used in connection with problems in the ventilation of mines, and of which some may be here given.

If p is the pressure per sq. ft., a the area of airway in sq. ft., s the total rubbing

surface of airway, v the velocity of air current in thousands of cub. ft. per minute, and K the coefficient of friction,*

$$\text{then (1) } pa = ksv^2, \quad \text{(2) } p = \frac{ksv^2}{a},$$

$$\text{(3) } a = \frac{ksv^2}{p}, \quad \text{(4) } k = \frac{pa}{sv^2},$$

$$\text{(5) } s = \frac{pa}{kv^2}, \quad \text{(6) } v^2 = \frac{pa}{ks}, \text{ or } v = \sqrt{\frac{pa}{ks}}$$

$$\text{while the H.P.} = \frac{ksv^3}{33000} \text{ (7).}$$

Practical considerations in reducing Friction.—*Reducing the Length.*—This can be most conveniently done when the airways are being constructed at first, and if the distance which the air current has to travel can in any way be reduced, even after the airways have been made, this ought to be done.

There are many advantages to be gained by having airways as short as possible, amongst which may be mentioned: The smaller cost and reduced expense for repairs and upkeep, and the larger volume of air which can be obtained by a given expenditure of power. All sharp angles should be avoided as much as possible.

Increasing the Area.—As to increasing the sectional area, there is a practical limit to which this can be done, depending largely on the nature of the strata through which the airways pass; for sometimes it is only with the greatest difficulty that a road can be enlarged, especially if the roof or floor is very bad, and if there is a continuous crush on the workings. For this reason it is often cheaper and easier to make an additional airway parallel to the one already existing.

By doing this the rubbing surface is doubled, and the resistance due to friction correspondingly increased; therefore the pressure will also require to be doubled. At the same time, however, doubling the rubbing surfaces has also doubled the *area*, and consequently the *velocity* will be *halved*, and the resistance and pressure reduced to one-fourth, the decrease of friction, owing to the reduced velocity in the larger area, being always much greater than the increase due to the extra rubbing surface.

Example.—If an airway measuring 9 ft. × 6 ft. is enlarged to 9 ft. × 8 ft., the area will be increased from 54 sq. ft. to 72 sq. ft., or in the ratio of 3 : 4, and the velocity will be reduced *inversely* in the same ratio, *i.e.* from 4 : 3, while the rubbing surface will, at the same time, be increased from 30 sq. ft. to 34 sq. ft., or in the ratio of 15 : 17. The resistance, however, varies as the square of the velocity, or as

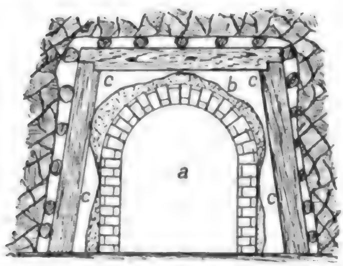
* For every foot of rubbing surface, and for a velocity in the air of 1000 ft. per minute, the friction is equal to 0·26881 ft. of air column of the same density as the flowing air, which is equal to a pressure, with air at 32° F., of 0·0217 lb. per sq. ft. of area of section. This is known as the coefficient of friction.

16 : 9, therefore the resistance, after increasing the area, will be reduced to $\frac{9}{16}$ of $\frac{17}{15}$ $(=\frac{153}{240})$, the net gain being over one-third.

Another important factor in limiting the area of airways is the size of the shafts, for with small shafts of insufficient size to permit of the passage of the requisite quantity of air, it is useless to increase the area of air-courses beyond a certain limit.

In making airways so as to present the least possible resistance the following points should be attended to. They should be as nearly as possible of the same sectional area throughout, have as few sharp angles as possible, and no side projections and narrow places in them. They should also have a good-sized area, and be as nearly circular, or, if this is impracticable, as square, as possible.

In experiments made recently by D. Murgue on the quantities of air passing in airways of varying construction, he found that there was a great deal of difference in the respective behaviour of arched airways, timbered and untimbered airways (fig. 436). In workings of the same area he found that the quantities passed would be 46,000 cub. ft. per minute for arched airways; 36,000 for airways without timber; and 32,000 for timbered courses, or in the proportion of $1\frac{7}{16} : 1\frac{1}{8} : 1$. Readings of the water-gauge for the same airways with these quantities were :—Arched airways, 0·253 in., airways without timber, 0·760 in., and airways with timber, 1·275 in., or in the ratio of 1 : 3 : 5 (approximately).



a = arched airway ; b = unlined airway ; c = timbered airway.

FIG. 436.*

These results show in a very striking manner the advantage of having good smooth passages for the air to travel in, and that often better results could be obtained by giving more attention to the airways than by trying experiments with better ventilating machines. For the different airways Murgue gives the following coefficients of friction, expressed as pressure per sq. ft. in decimals of a lb. for each square foot of rubbing surface and an air velocity of 1000 ft. per minute :—

Arched	passages,	average	coefficients	·001716
Unlined	,,	,,	,,	·00494
Timbered	,,	,,	,,	·00821

The laws of ventilating may be summed up and briefly stated as follows :—

* The illustration is intended to show graphically that the arched airway will pass as much air with the same ventilating pressure as either the unlined or timbered airways described on its outside.

The quantity of air circulating in a mine is as the square root of the pressure or the square root of the water-gauge reading.

In airways of the same sectional area, and which vary in length only, the volume and velocity of air currents are *inversely* proportional to the square root of their lengths.

The quantity of air passing in airways of different areas is, other things equal, in proportion to the square root of the area multiplied by the area itself.

The resistance varies directly as the length.

The pressure required to propel air through airways is inversely proportional to the areas, other conditions remaining the same.

The quantity of air circulating is proportional to the cube root of the power applied.

Since the quantity of air circulating varies as the cube root of the power applied, and as the number of revolutions of a fan vary in the same ratio, it follows that the quantity of air circulating at any moment depends *directly* on the speed of the fan.

Other general formulæ, as arranged by Merivale, may here be given :—

a. Ventilating pressure varies as depth of up-cast shaft (in furnace ventilation).

b. Ventilating pressure varies as difference of temperature between up-cast and down-cast shafts.

c. Ventilating pressure varies as horse-power of fan or furnace.

d. Ventilating pressure varies as quantity of coal burned.

e. Quantity of air circulating varies as revolutions of fan.

f. Quantity of air circulating varies as square root of pressure.

g. Quantity of air circulating varies as square root of depth of up-cast (in furnace ventilation).

h. Quantity of air circulating varies as difference of temperature between up-cast and down-cast shafts.

i. Quantity of air circulating varies as cube root of horse-power.

j. Quantity of air circulating varies as cube root of coals burned.

'Splitting' the Air.—The benefits to be derived by increasing the area of airways, and also by adding another airway, have been already pointed out. The latter method of solving this problem is termed 'splitting the air.' In former times (and even in some collieries at the present day) it was the practice to carry all the air round the workings in one current, which is a very bad method, and one that should be avoided if possible. The air should be taken round the workings in a number of separate currents, as only by this system can the greatest efficiency be obtained.

Splitting the air, as already shown, increases the rubbing surface, but it reduces the velocity with the same pressure, and as the pressure varies as the rubbing surface and the velocity squared, a greater quantity of air can be propelled by a given power.

There are other advantages gained by splitting the air, and ventilating each section separately with main splits, among which are the following :—

A greater total quantity of air is got.

By splitting the air each section gets a more uniform and purer supply.

A fall occurring in one split or section does not injure the others.

An explosion taking place in one section is less liable to occasion disastrous effects in the other sections.

Fewer trap-doors are required on the main roads.

To obtain the greatest possible advantage from 'splitting,' the main splits ought to commence as near the down-cast and end as near the up-cast shafts as possible.

In 'splitting,' the main current of air should never be split to such an extent that its velocity becomes insufficient to keep the working faces clear of gas. If this be the case, fire-damp or choke-damp may accumulate, and render the working-places dangerous.

The velocity of the air travelling along the faces in fiery mines where unshielded lamps are used should never be less than 120 to 150 ft. per minute, nor more than 350 ft. per minute.

In ordinary circumstances, the velocity along the face should be from 2 to 3½ ft. per second, in splits 3½ to 5 ft. per second, in main airways 5 to 10 ft. per second, and in shafts it often reaches 20 to 40 ft. per second.

The relative proportion of air to be distributed among any given number of splits will depend greatly on the amount of natural gases given off, apart from the actual quantity required for men and animals, the burning of lamps and the blasting of explosives, so that in distributing a current it is best to divide it according to local circumstances; *i.e.* the section giving off the most gas should get the largest supply of air. The average quantity allowed by some authorities for mines free from fire-damp is 150 to 200 cub. ft. per minute per man and boy, and three to six times these quantities for each horse employed underground. In fiery collieries 200 to 350 cub. ft. per minute per man and boy ought to be allowed, and an additional quantity for 'scale,' as a certain proportion never reaches the working faces, being lost by leakage through defective stoppings, bratticing, and screens. The above quantities are only approximate.

For a fiery mine employing 200 men and 20 horses, the quantity required would be:—

men $200 \times 350 = 70{,}000$ cub. ft.
horses $20 \times 2100 = 42{,}000$,, $= 112{,}000$ cub. ft. per minute.

Equal Splits. — The expression 'two equal splits' means that the original single air-current has been divided into two currents, and each current traversing an airway of *half* the original length, but both with the *same area* as the original one. 'Equal splitting' is more of a theoretical than a practical expression, for to split an air current into two or more equal volumes can rarely be carried out in practice, although in some cases it may approximately be done.

When the original single current is divided into two equal splits, they may be considered as one current with *double* the area, but with the *same* rubbing surface; likewise, when divided into three equal splits, they may be considered as one with

three times the original area but with the same rubbing surface. This may be illustrated as follows:—

	Area.	Rubbing Surface.
Original airway, . .	50 sq. ft.	160,000 sq. ft.
Two equal splits, .	50 ,,	80,000 ,,
	50 ,,	80,000 ,,
	100 ,,	160,000 ,,
Three equal splits, .	50 ,,	53,333·3 ,,
	50 ,,	53,333·3 ,,
	50 ,,	53,333·3 ,,
	150 ,,	160,000 ,,

Example.—If the total quantity of air passing round the workings of a mine is 20,000 cub. ft. per minute when the size of the airway is 8 ft. × 5 ft. and 2500 ft. long; what quantity will circulate if the current is divided into 2, 3, and 4 equal splits?

First find the ventilating pressure required to circulate the original quantity through the given airway, by the formula

$$p=\frac{ksv^2}{a}=\frac{\cdot 0217\times(26\times 2500\times\left(\frac{20,000}{1000\times 40}\right)^2}{40}=8\cdot 8 \text{ lbs. per sq. ft.}$$

To find quantity with new area, $v=\sqrt{\frac{pa}{ks}}\times a_2$, but p, k and s are the same in each instance, therefore the quantity passing in two equal splits will be as $\sqrt{a_1}\times a_1 : \sqrt{a_2}\times a_2 :: Q : x$

$$\therefore x=20,000\,\frac{\sqrt{80}\times 80}{\sqrt{40}\times 40}$$

$$=56,582 \text{ cub. ft. per minute.}$$

The same result approximately may be arrived at by simply taking the number n_1 of splits, thus $\sqrt{1}\times 1 : \sqrt{2}\times 2 :: 20,000 : x$

and
$$x=20,000\,\frac{2\sqrt{2}}{1\sqrt{1}}$$

$$=20,000\times 1\cdot 4142$$
$$=56,568 \text{ cub. ft.}$$

For three equal splits the quantity $x=20,000\,\frac{3\sqrt{3}}{1\sqrt{1}}=103,920$ cub. ft.

and for four equal splits the quantity $x=20,000\,\frac{4\sqrt{4}}{1\sqrt{1}}=160,000$ cub. ft.

Unequal Splitting.—This expression is used when an air current is split up into a number of separate currents each of which traverses an airway of different dimensions and therefore meets with a varying amount of resistance.

If there are a number of splits of *unequal* area, subject to a common pressure, the quantities of air that will pass in each are in proportion to

$$\frac{1}{\sqrt{\dfrac{\left(\frac{1}{a}\right)^2\times S}{a}}} \text{ or } R=\sqrt{\frac{a^3}{S}}.$$

where R = relative quantity of air passing into each airway.
s = total rubbing surface in sq. ft. of each airway.
a = area of each airway in sq. ft.

Example.—An airway 7 ft. × 9 ft. × 1200 ft., through which 80,000 cub. ft. per minute is passing, is divided into three unequal splits of the following dimensions: first split, 6 ft. × 7 ft. × 1200 ft.; second split, 6 ft. × 6 ft. × 900 ft.; third split, 6 ft. × 4 ft. × 840 ft. With the same total volume, what quantity will pass in each airway?

Let R_1, R_2, and R_3 denote the relative quantities going into each airway, then by the formula $R=\sqrt{\frac{a^3}{s}}$,

$$R_1=\sqrt{\frac{(6\times7)^3}{(26\times1200)}}=\sqrt{\frac{74,088}{31,200}}=1{\cdot}54.$$

$$R_2=\sqrt{\frac{(6\times6)^3}{(24\times900)}}=\sqrt{\frac{46,656}{21,600}}=1{\cdot}47.$$

$$R_3=\sqrt{\frac{(6\times4)^3}{(20\times840)}}=\sqrt{\frac{13,824}{16,800}}=0{\cdot}90.$$

These relative quantities show that for every 1·54 cub. ft. going into the first split, 1·47 cub. ft. will be going into the second, and 0·90 cub. ft. going into the third split.

Let the total relative quantity be denoted by R_4,

then
$$\begin{aligned} R_4 &= R_1 + R_2 + R_3 \\ &= 1{\cdot}54+1{\cdot}47+0{\cdot}90 \\ &= 3{\cdot}91 \end{aligned}$$

The total volume x_1 going into the first split will be as

$$R_4 : R_1 :: Q : x_1$$
$$3{\cdot}91 : 1{\cdot}54 :: 80,000 : x_1$$

$$\therefore x_1=\frac{80,000\times1{\cdot}54}{3{\cdot}91}=31,508 \text{ cub. ft. per minute.}$$

For second split

$$3{\cdot}91 : 1{\cdot}47 :: 80,000 : x_2$$

$$\therefore x_2=\frac{80,000\times1{\cdot}47}{3{\cdot}91}=30,076{\cdot}72 \text{ cub. ft. per minute.}$$

For third split

$$3{\cdot}91 : 0{\cdot}90 :: 80,000 : x_3$$

$$\therefore x_3=\frac{80,000\times0{\cdot}90}{3{\cdot}91}=18,416 \text{ cub. ft. per minute.}$$

$$31,508+30,076+18,416=80,000 \text{ cub. ft.}$$

The relative pressure or power required to pass the same quantity of air through airways of varying area and length may be found by the formula

$$\frac{S\left(\frac{1}{a}\right)^2}{A} \text{ or } S\left(\frac{1}{a}\right)^3 \text{ where}$$

A = area of roadways in sq. ft. S = Total rubbing surface in sq. ft.
a = ,, ,, R = Relative quantity.

Instruments used at Collieries.—*Thermometer.*—A thermometer consists of a closed glass tube, with a capillary bore in the upper part, and a bulb below containing mercury or spirits of wine, and provided with a graduated scale having two fixed points, viz., freezing point and boiling point. There are three different kinds of thermometric scale adopted for recording the temperature, viz., the Fahrenheit, Centigrade, and Réaumur.

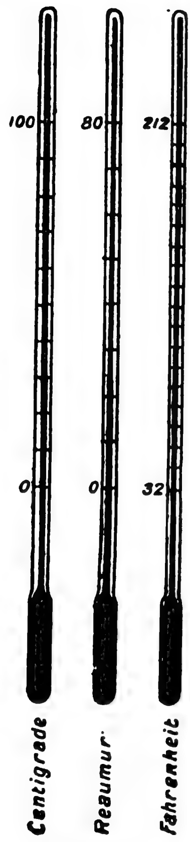


FIG. 437.

Réaumur divided the space between the freezing and boiling points into 80 equal parts, but there is no good reason why he should have preferred 80 over any other whole number. This scale is much used in Italy, Russia, and some parts of Germany. Celsius, a Swede, divided his thermometer scale into 100 equal parts between the boiling and freezing points: his scale is known as the Centigrade, and it is used in France and other countries of Europe, and is also being largely used in Britain among scientific men.

In Fahrenheit's scale the space between freezing and boiling points is divided into 180 equal parts, and he fixed his freezing point 32° above zero. This scale is the one most commonly used in Britain.

By the use of this instrument we are able to measure the temperature of the air in the workings, or note the difference in temperature between the air in the up-cast and down-cast shafts.

The Barometer is an instrument used for measuring the pressure of air. Its construction is as follows: A glass tube 36 in. long, closed at one end, is filled with mercury and reversed, the open end being temporarily closed until it is placed so that it shall project below the surface of a bath of mercury contained in a reservoir. The barometer has a scale fixed to it, and also a sliding vernier, by means of which it may be read to the one-hundredth part of an inch. Each inch on the scale is divided into tenths, and the divisions on the vernier are one less than the number on a corresponding length on the barometer scale.

A barometer is of great use at collieries, as it shows the changes in atmospheric pressure. With a low barometer the gas issues more freely from the coal, and if there is any gas lying in the waste it will soon find its way into the working places, owing to the reduced

pressure. It is important, therefore, to note the rise or fall of the mercury daily in connection with fiery collieries.

By the Coal Mines Regulation Act every mine must be provided with a barometer and a thermometer. The reading of the former should be noted at least once a day, especially by the firemen before going down to inspect the workings in the morning.

Water-Gauge.—A current of air exerts a certain pressure during its passage from one point to another, which pressure is usually measured by a water-gauge (fig. 438).

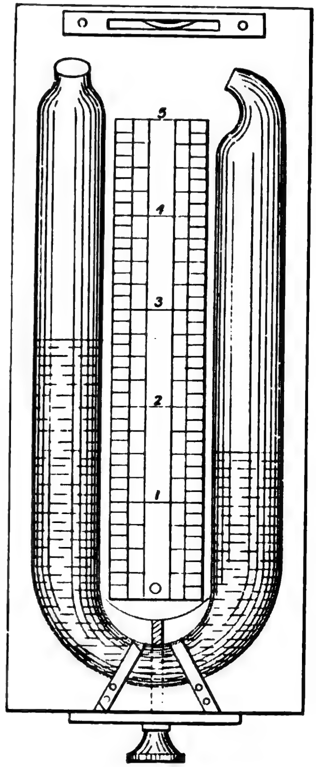


FIG. 438.—Water-gauge.

The construction of this instrument is very simple. It consists of a glass U-tube, of $\frac{1}{2}$ in. to 1 in. in diameter, to which a sliding scale divided into inches and tenths is attached. One of the ends is fitted with a tube at a right angle with the limb, which can pass into the intake through a screen, while the other end is connected with the return current. If there is any difference in the pressure between the two currents the water level in the two limbs of the gauge will vary. As 1 cub. ft. of water weighs 62·5 lbs., a cub. in. will weigh ·036 lb. In a tube of 1 in. sectional area, a difference of level of 1 in. will represent a wind pressure equal to the weight of 1 cub. in., and, *pari passu*, to ·036 lb. per sq. in., or $\cdot036 \times 144 = 5\cdot2$ lbs. (approximately) per sq. ft.

The water-gauge thus acts as a check (though not altogether a very reliable one) on the state of the underground airways. If the latter remain in the same condition for some time, and the ventilating power is neither increased nor decreased, then under ordinary circumstances the height of the water-gauge should not vary. If the difference in level is very much greater than usual, it will probably be due to the fact that the air is not taking its usual course, owing to some doors being open in the main airways, by which the air will be going direct from the down-cast to the up-cast without ventilating the workings. If the water-gauge is to be of any use at all, it ought to be placed as far back from the fan as possible, otherwise it will be

impossible to get correct results, owing to the abnormal condition of the air close to the fan.

Hygrometer.—The amount of moisture in the air can be ascertained by the use of the instrument known as the hygrometer. It consists of two thermometers (fig. 439) mounted at a short distance from each other, the bulb of one being covered with muslin, which is kept moist by means of a cotton wick dipping into a vessel of water. The evaporation which takes place from the moistened bulb produces a depression of temperature, so that this thermometer gives a lower reading than the other by an amount which increases with the moisture of the air. The instrument must be mounted in such a way that the air can circulate freely round the wet bulb.

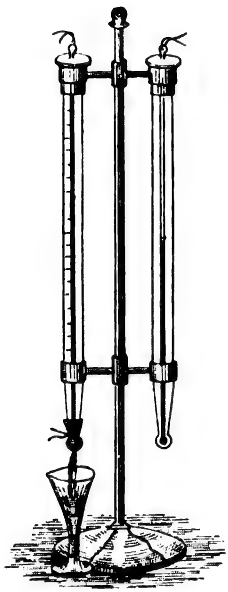


FIG. 439.—Hygrometer.

By means of a formula,* the tension due to the vapour of water in the air is calculated from the readings of the two thermometers. Tables have also been constructed, by means of which the degree of saturation can be calculated.

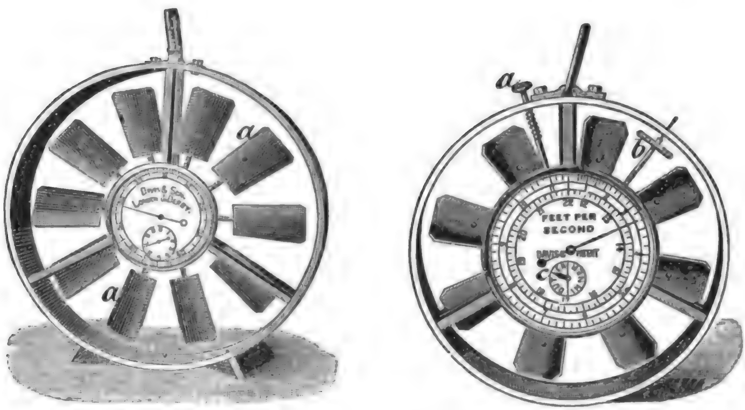
Anemometers.—To determine the velocity of the air passing in underground workings, it is the common practice to use the instrument called the anemometer. It is constructed somewhat like a small fan, with a number of blades or vanes fixed obliquely to the axis (see figs. 440, 441); when these vanes are rotated a clockwork gearing connected to the axis, which actuates the pointers on the dial of the instrument, records, by means of a scale, the velocity in feet at which the air is travelling. In determining the quantity of air passing, the pointers are first brought to zero, and the anemometer is held out at arm's length in the air passage for a measured period of time, and a reading is taken, the hundreds being read off the small pointer, and the odd feet read off the large one. The sectional area of the passage is carefully calculated, and the total quantity of air passing can be ascertained by multiplying the velocity per minute by the area in sq. ft.

Coal-Dust and Methods of dealing with it.—That coal-dust is an important element in connection with explosions in underground workings seems to be fully recognised by all or nearly all connected with mining operations. It is many years since this fact was first pointed out and experiments made by two eminent authorities, Faraday and Lyell. It is over fifty years ago since these distin-

* See Deschanel's ***Natural Philosophy***, p. 398.

guished men first gave their opinion on this much-debated question, but it is only within recent years that much attention has been given to the subject. To Mr William Galloway and the late Sir Frederick Abel we are greatly indebted for the information we possess on the subject. These two authorities studied the question very closely for years, and after many careful and comprehensive experiments, came to the conclusion that "coal-dust is highly dangerous under certain conditions." Others have entered the field of investigation, and the results have placed beyond doubt the statement that coal-dust, rather than fire-damp, plays the most important part in many colliery explosions.

There are two theories held by mining authorities regarding the action of coal-dust in colliery explosions, one being that coal-dust alone will cause an explosion without any fire-damp being present in



Figs. 440, 441.—Anemometer.

the air, and the other, that before coal-dust becomes really dangerous a certain percentage of fire-damp must be present in the air.

We may here quote briefly the opinion of different authorities who were commissioned to make inquiry on the subject. MM. Mallard and Le Chatelier of the French Fire-damp Commission of 1882 rejected the theory that coal-dust alone would cause any serious danger, or that any colliery explosion of importance could be attributed, with any probability of authenticity, to the action of coal-dust.

The Prussian Fire-damp Commission in 1887 came to the conclusion that the presence of coal-dust in the complete absence of fire-damp gave rise generally to an elongation or propagation of the flame projected by a blown-out shot of limited extent, however far the deposits of dust may extend in the mine roads, but that there were certain descriptions of coal-dust which, if ignited by a blown-out shot,

would not only continue to carry on the flame, even to distances much beyond the confines of the dust deposits, but would also, in the entire absence of fire-damp, give rise to explosive results, which, in character and effects, were similar to those produced with some other dusts in air containing 7 per cent. of fire-damp.*

The Austrian Fire-damp Commission in 1891, after making a large number of experiments with different coal-dusts, found that, in the absence of fire-damp, nearly all kinds of coal-dust could be ignited by a 3½-oz. dynamite cartridge exploded in an unconfined space, while many dusts which were notoriously regarded as dangerous proved less inflammable than others which had been regarded as comparatively innocent, the fineness of the dust greatly increasing the liability to ignition.

The Royal Commission on Accidents in Mines, in their report of 1886, were satisfied that "a blown-out shot in working-places *where highly inflammable coal-dust exists in great abundance*, may, *even in the entire absence of fire-damp*, possibly give rise to a violent explosion, or may, at any rate, be followed by the propagation of flame through very considerable areas, and even by the communication of flame to distant parts of the workings where explosive gas-mixtures or dust deposits in association with non-explosive gas-mixtures exist."

The Royal Commission on Explosions from coal-dust in mines issued a further report in 1894, in which they came to the following conclusions:—

(1) The danger of explosion in a mine in which gas exists, even in very small quantities, is greatly increased by the presence of coal-dust.
(2) A gas explosion in a fiery mine may be intensified and carried on indefinitely by coal-dust raised by the explosion itself.
(3) Coal-dust alone, without the presence of any gas at all, may cause a dangerous explosion if ignited by a blown-out shot or other violent inflammation. To produce such a result, however, the conditions must be exceptional, and are only likely to occur on rare occasions.
(4) Different dusts are inflammable, and consequently dangerous, in varying degrees, but it cannot be said with absolute certainty that any dust is entirely free from risk.
(5) There appears to be no probability that a dangerous explosion of coal-dust alone could ever be produced in a mine by a naked light or ordinary flame.

Sir Frederick Abel states that it requires 2 to 2½ per cent. of fire-damp, added to a mixture of coal-dust and air, to make it explosive, while Mr William Galloway says that 1 per cent. of fire-damp in a mixture of fine coal-dust and air is sufficient to form an explosive mixture, the quantity of dust constituting a danger being 1 lb. to 160 cub. ft. of air.

These authorities have been quoted to show the difference of opinion that exists in regard to this important matter. Since these reports were issued, conclusive evidence has been forthcoming that an

* *Final Report of the English Commission on Accidents in Mines*, 1886.

explosion may take place through the agency of coal-dust alone, as such an explosion, attended with loss of life, actually took place at Camerton Colliery in Somersetshire, where fire-damp was not known to have been present prior to the explosion. In this instance it was clearly proved that the explosion was caused by a blown-out shot fired by two men who were repairing on the main haulage road, which was dusty.

At the West Riding Colliery in Yorkshire an explosion took place which was proved, after careful inquiry, to have been caused by coal-dust alone, ignited by a blown-out shot.

Before an explosion can take place, certain conditions, which are fortunately of rare occurrence in mines, must be fulfilled.* There must be sufficient surplus power in the shot to stir up the dust and to produce sufficient compression of the air. The dust must be very fine and dry; the flame from the explosive must be one of very high temperature and considerable volume. Possibly the shape of the gallery, the relative temperature of the air, and the height of the barometer, may also conduce to modify or intensify the effects. From a consideration of the composition of coal-dust, it is evident that it is a very inflammable substance. Analyses of coal-dust from Ryhope Colliery showed on an average 21·17 per cent. of combustible gas. Dust from Brancepeth Colliery,† however, only gave 0·76 per cent. of combustible gas, and this latter dust was, notwithstanding the small percentage of combustible gas with which it was associated, very sensitive to ignition.

Coal-dust is prevalent to a greater or less extent in all collieries, particularly in deep and dry ones. It is more frequent in deep mines, because the temperature of the air is higher in such mines than in shallow ones, and the air being warmer, the moisture in the atmosphere, as it enters the workings, is absorbed very rapidly, causing the air to become dry and favouring the formation of dust. Some seams, again, produce more dust than others, and the more friable the coal the more dust will accumulate.

Formation of Dust.—Coal-dust is most plentiful on main haulage roads, and is found settled on the roof, floor, sides, and timbers. The dust is formed, and accumulates in a variety of ways. Thus the 'backs' or cleavages of the coal are usually coated with a fine, friable, black substance, which becomes dust when the coal is broken up by the miner at the face, while the hewing and filling of the coal also contribute largely to the formation of dust. The breaking-off of pieces of coal from the sides of pillars in pillar and stall working, and the falling coal from tubs during haulage, and the subsequent pulverisation it is subjected to by the feet of men and horses, and by the wheels of the tubs passing along the roads at a rapid speed, and the descent of large quantities of very fine dust from the screens on

* *Trans. Fed. Inst. Min. Eng.*, vol. vii. p. 60.
† *Trans. Fed. Inst. Min. Eng.*, vol. vii. pp. 33–38.

the surface along with the air current descending the down-cast shaft, also occasion the deposition of dust.

Methods of dealing with Coal-Dust.—Some means must be adopted to deal with and render less dangerous these accumulations of dust, especially if fire-damp is given off, and shots are being fired in the mine. Coal-dust may be dealt with by different methods, such as :—

(*a*) By removing it altogether ; (*b*) damping it with salt or other hygroscopic substances ; (*c*) damping it with water.

Removing the dust altogether is impracticable in most collieries, on account of the time, labour, and expense it would involve. The dust that adheres to the sides, roof, and timber is usually the most dangerous because of its fineness, and even sweeping with a brush would not wholly remove it. Clearing part of the roadway, if this can be thoroughly done, and keeping it clean and well watered, may have some effect in checking the course of the flame if an explosion should occur.

Damping the dust with salt is carried out to a considerable extent, and is found, in many cases, to give better results than watering, but is much more expensive. Salt water has also been used instead of pure water and gives good results, as it keeps the dust in a moist condition longer, and also tends to form a deposit over the particles. Water containing clay has been tried in preference to pure water, as it has a binding or caking action somewhat similar to that of salt water, but is neither so easily prepared nor used.

Damping by water is the method that is commonly used in most collieries. The method of watering may be either by watering-carts, by automatic watering-tanks, or by pipes. Sometimes when a watering-cart is used it is provided with a revolving brush, driven from the axles, which sprays the water all round, sprinkling the roof, floor, and sides, and is much more effective than the common watering-cart (see fig. 442). Automatic tanks, with pumps attached, and worked by cranks from the axle, have also been used. Watering by means of pipes is, however, the method that is most in use, and is no doubt the most effective.

A simple and serviceable arrangement is to lead a line of pipes about 1½ in. or 2 in. diameter along the side of the road, and have taps which can be connected to a rubber hose with a 'rose' set so as to allow the water to escape in a spray, placed at intervals of 40 or 50 ft. The rubber hose should be 20 or 25 ft. long, so that a length of 40 to 50 ft. may be watered each time it is connected to the water pipes. The supply of water is usually obtained by connecting the pipes to the pumping set in the shaft, or to some tank placed so as to afford the required pressure.

In some collieries the water is allowed to issue from small jets in the form of very fine spray, at high pressure, in the intake air currents, and is carried along to moisten the floor, roof, and sides. This method has the further advantage of keeping the air cool,

and does not injure the floor so much as watering by a water-cart would do.

Water combined with compressed air has also been successfully employed for laying the dust, and is also effective in keeping the air cool. What is known as Martin and Turnbull's system, which is extensively used in the South Wales coal-field, is probably the best method of applying compressed air and water. Where compressed air is used for power purposes in the mine, the air in the spray-producer is taken from the compressed air main by a ½-in. wrought-iron tube to the producer, which is generally fixed in the centre of the roadway. A water main is carried parallel with the air main, and from it to the same producer another ½-in. tube

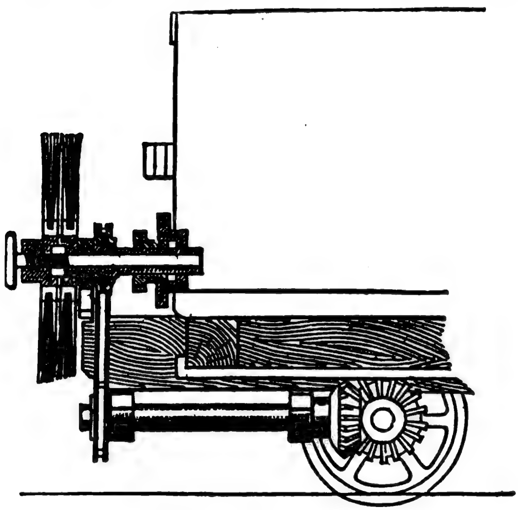


FIG. 442.—Water-cart.

conveys the water. Immediately before entering the spray-producer the air and water pipes are united. The air passes through a conical nozzle, whilst the water issues through a similar orifice around the conical water nozzle, where they are united in one stream.

Fig. 443 shows the arrangement of this appliance in elevation. The compressed air main A, and the water main pipe B, are laid parallel with one another, preferably at one side of the road. At any desired intervals in these mains, branch pipes *a* and *b* are carried up towards the roof. By these pipes the compressed air and water are carried into the delivery pipe C, into which the air pipe *a* extends in the form of a nozzle, terminating beyond the point of junction of the water pipe *b*, with the coupling *d*, as shown in figure.

The air and water are discharged into the mine through a terminal

instrument or nozzle, so constructed as to cause the issuing compressed air and water to form a fine spray or cloud of vapour. Another convenient form of spray-producer is composed of two cup-shaped members, of which one is inverted on the other. This cup is connected by screw-threads to the end of the delivery pipe, into which the compressed air and water are received from the pipes.

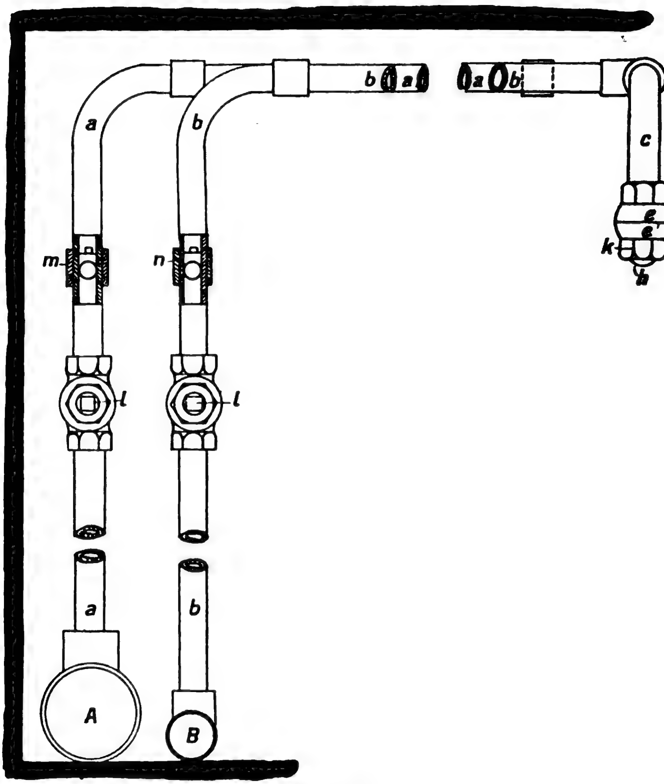


FIG. 443.

The air and water pass by the orifices into the cavity formed by the cups. Extending downwards from the top cup is a spindle, the lower end of which is furnished with screw-threads. The lower cup fits loosely over the lower end of the spindle, and a washer is interposed between the cup and a shoulder formed on the spindle. By a nut working on the screwed part of the spindle, the cups are held towards each other, and the degree of fineness of the spray is governed by the extent to which the lower cup is screwed towards

the upper one. The degree of fineness of the spray escaping between the rims of the cups is capable of the nicest adjustment, and, when required, the issue of the spray may be made to cease by forcing the lower cup sufficiently tight against the upper one. With this form of spray-producer a sufficiently fine spray may be obtained when water alone is used. Another form of spray-producer consists of a pipe, the end of which is flattened so as to present a narrow slit or orifice for the issue of the compressed air and water. The pressure of the water is in excess of that of the compressed air, and the supply of each is controlled by taps situated in the branch pipes. In the event of a cessation of the supply of compressed air, the water is prevented by a valve from passing into the main pipe; and similarly, should an accident happen to the water main, the compressed air is prevented by a valve from passing into it. By this system the extreme fineness of the spray is such that it is carried by the ventilating current in the mine for long distances, and effectually cools the air in the mine and damps the dust lurking in crevices and behind the timber, as well as on the roof, floor, and sides, without unnecessarily wetting the roads, which so often causes creep in the floor.

From investigations made in Germany on the subject of watering coal-dust, it would seem that the mere spraying of the dust, while it may improve the climatic conditions of the underground workings by cooling the atmosphere, would not greatly diminish the risk of fire-damp explosions. Samples of coal-dust taken, after watering, at Maybach Colliery, showed 2·05 to 3·65 per cent. of water, but the researches of the Prussian Fire-damp Commission have demonstrated that the explosive properties of coal-dust are not destroyed until it has taken up 50 per cent. of its weight in water. From this it would seem that to wholly prevent coal-dust explosions nothing short of systematic and thorough watering will be effective.

Where dust is present in dangerous quantities, and especially in fiery collieries, as little blasting as possible should be allowed. The explosive used should be of the non-flaming type, such as carbonite or roburite, and the Coal Mines Regulation Act (section 49, 10) carried out with strict observance. Testing the return air with some lamp more sensitive than the ordinary 'Davy' lamp should also be a frequent precaution in dusty and fiery collieries. In collieries where the 'Davy' lamp is used to examine the workings for fire-damp, there must be at least from 3 per cent. to 4 per cent. of this gas present in the air before it can be detected, and, as has already been seen, both Mr William Galloway and Sir Frederick Abel are of the opinion that 1 per cent. to 2 per cent. of fire-damp present in the air, and coal-dust plentiful in the workings, might cause a disastrous explosion.

Wherever a coal seam that gives off fire-damp to any extent is worked, the air in the workings must always contain a greater or less

proportion of this gas, no matter how efficient the ventilation may be. Special precautions should always be taken under such circumstances, and an examination made with a Mueseler lamp, burning spirit, or some other good fire-damp detector.

The following problems are given to further illustrate the application of some of the formulae given in the foregoing chapter on ventilation :—

Example. — Two airways, one 10 ft. × 5 ft. × 300 fms. long, and the other 4 ft. × 6 ft. × 500 fms. long, circulate a total quantity of 30,000 cub. ft. per minute. What quantity of air would at a pressure of 1 lb. per sq. ft. at the down-cast shaft pass into each under the same conditions?

Let the 10 ft. × 5 ft. airway be designated x and the other y.

In the case of x

$$PA = KSv^2$$
$$1 \times 50 = \cdot01 \times 30 \times 1800 \times v^2$$
$$1 = 108\ v^2$$
$$\therefore v^2 = \frac{1}{108}, \text{ and } v = \sqrt{\cdot0925} = \cdot390$$

Now $V = \cdot390 \times 1000 = 390$ ft. per minute

And $Q = V \times A = 390 \times 50 = 19{,}500$ cub. ft. per minute

(Quantity of air circulating in x).

$$PA = \frac{y}{KSv^2}$$
$$1 \times 24 = \cdot01 \times 20 \times 3000 \times v^2.$$
$$1 = 25\ v^2$$
$$\therefore v^2 = \frac{1}{25} \text{ and } v = \sqrt{\cdot04} = \cdot200$$

$V = \cdot200 \times 1000 = 200$ ft. per minute

$Q = V \times A = 200 \times 24 = 4800$ cub. ft. per minute

(Quantity circulating in y).

$$\therefore x + y = 19500 + 4800 = 24300 \text{ cub. ft.}$$

circulating with a pressure of 1 lb.

But the total quantity circulating in both airways was 30,000 cub. ft. per minute.

$\therefore$ (by proportion) 24,300 : 30,000 : : 4800 : y

$$\text{and } y = \frac{30{,}000 \times 4800}{24{,}300} 5925 \cdot 92 \text{ cub. ft. per minute,}$$

and $30{,}000 - 5925 \cdot 92 = 24{,}074 \cdot 08$ cub. ft. per minute = quantity passed by x.

Example.—If two airways of the same area pass a total quantity of air equal to 10,000 cub. ft. per minute, subject to the same pressure in each case, the resistance in the airways being in the proportion 5 to 1, what proportion would each airway pass respectively?

This problem may be worked out in same way as the preceding one, or by the formula represented.

$$R = \sqrt{\frac{A^3}{S}}, \text{ where R is the relative quantity going into each.}$$

Let the two airways be represented by x and y, and let A equal 1 in both cases.

$$\text{Then } R = \sqrt{\frac{A^3}{S}} = \sqrt{\frac{1^3}{1}} = 1 = \text{relative quantity in } x,$$

$$\text{and } R = \sqrt{\frac{1^3}{5}} = \cdot447 = \text{relative quantity in } y.$$

The actual quantities passing into x and y will therefore be found by proportion,

$$\text{thus } 1\cdot447 : \cdot447 :: 10000 : y = \frac{\cdot447 \times 10{,}000}{1\cdot447} = 3089 \cdot 15 \text{ cub. ft. per minute,}$$

$$\text{and } x = 10{,}000 - 3089 \cdot 15 = 6910 \cdot 85 \text{ cub. ft. per minute.}$$

Example.—Find the total quantity of air per minute passing in an airway 10 ft. × 7 ft. × 2000 ft., at a pressure of 8·5 lbs. per sq. ft.

$$PA = KSv^2.$$

Here $P = 8{\cdot}5$ lbs.; $A = 70$ sq. ft.; $Per = 34$; $S = 34 \times 2000$; $v^2 = \left(\frac{Q}{1000 \times A}\right)$

$$8{\cdot}5 \times 70 = {\cdot}01 \times 34 \times 2000 \times v^2$$

$$595 = 680v^2$$

$$\therefore v^2 = \frac{595}{680}; \text{ and } v = \sqrt{{\cdot}8750} = {\cdot}9354,$$

and Q the total quantity $= {\cdot}9354 \times 1000 \times 70 = 65{,}478$ cub. ft. per minute.

Example.—If the quantity of air passing round a mine is 10,000 cub. ft. per minute before splitting, when the size of airway is 6 ft. × 5 ft. × 1200 ft. long, what quantity will circulate if the current is split respectively into two, three, and four equal parts, the pressure and other conditions remaining the same?

Here the pressure and rubbing surface will be the same in each case, therefore the quantity will vary as $\sqrt{A_3}$ or $Qn = \sqrt{n} \times n \times Q$.

(1) $Qn_1 = \sqrt{2} \times 2 \times 10{,}000 = 28{,}280$ cub. ft. in two equal splits.

(2) $Qn_2 = \sqrt{3} \times 3 \times 10{,}000 = 51{,}930$ cub. ft. in three equal splits.

(3) $Qn_3 = \sqrt{4} \times 4 \times 10{,}000 = 80{,}000$ cub. ft. in four equal splits.

Example.—A down-cast shaft, 14 ft. diameter and at a temperature of 50° F., passes 121,000 cub. ft. per minute. What size of shaft would the up-cast require to be, if the velocity of air current in both shafts is to be equal, supposing the temperature in the up-cast is 100° F.?

$$\text{Velocity in down-cast} = \frac{Q}{A} = \frac{121{,}000}{14^2 \times {\cdot}7854} = 786{\cdot}1 \text{ ft. per minute.}$$

$$\text{Increase of volume in up-cast} = \frac{Q \times t}{459} = \frac{121{,}000 \times 50}{459} = 13{,}180{\cdot}82 \text{ cub. ft.}$$

$$\therefore \text{total volume in up-cast} = 121{,}000 + 13{,}180{\cdot}82 = 134{,}180{\cdot}82 \text{ cub. ft.}$$

$$\text{and area of up-cast} = \frac{Q}{V} = \frac{134{,}180{\cdot}82}{786{\cdot}1} = 170{\cdot}69 \text{ sq. ft.}$$

$$\therefore \text{diameter of up-cast} = \sqrt{\frac{170{\cdot}69}{{\cdot}7854}} = 14{\cdot}74 \text{ ft.}$$

Example.—Two shafts, each 15 ft. × 5 ft. × 100 fms. deep, are connected by a drift 11 ft. × 5 ft. × 400 yds. long, the quantity of air passing being 30,000 cub. ft. per minute. Find the quantity that would pass if another drift were added 11 ft. × 5 ft. × 400 yds. long, (1) with same pressure, (2) with same horse-power?

(1) The pressure required will be the same for each shaft.

$$\text{Now } P \times A = K \times S \times v^2$$

$$\therefore P \times 15 \times 5 = {\cdot}01 \times 40 \times 600 \times \left(\frac{30{,}000}{75 \times 1000}\right)^2$$

$$\therefore P = {\cdot}01 \times 8 \times 40 \times \left(\frac{2}{5}\right)^2 = {\cdot}51 \text{ lb. per sq. ft.}$$

and the pressure in each shaft $= {\cdot}51 \times 2 = 1{\cdot}02$ lbs.

(2) Find pressure in drift. $P \times 55 = {\cdot}01 \times 32 \times 1200 \times \left(\frac{30{,}000}{55 \times 1000}\right)^2$

$$P \times 11 \times 121 = {\cdot}01 \times 32 \times 240 \times 36$$

$$\therefore P = \frac{{\cdot}01 \times 32 \times 240 \times 36}{11 \times 121} = 2{\cdot}07 \text{ lbs. per sq. ft.}$$

If another drift of the same size were added, the velocity would be halved, and therefore would only require ¼ the pressure and $\frac{2 \cdot 07}{4} = \cdot 517$ lb. to pass 15,000 cub. ft. in each airway.

Pressure in shaft and drift before splitting $= 2 \cdot 07 \times 1 \cdot 02 = 3 \cdot 09$ lbs.

,, after adding an additional airway $= 1 \cdot 02 \times \cdot 517 = 1 \cdot 537$ lbs.

Now quantity $\propto \sqrt{\text{pressure}}$. $\therefore \sqrt{1 \cdot 537} : \sqrt{3 \cdot 09} :: 30{,}000$ cub. ft.

$\therefore 1 \cdot 23 : 1 \cdot 75 :: 30{,}000 : x$

and $x = \frac{30{,}000 \times 1 \cdot 75}{1 \cdot 23} = 42{,}482 \cdot 92$ cub. ft. per minute (when the second drift is added).

$$\text{H.P. before adding airway} = \frac{30{,}000 \times 3 \cdot 09}{33{,}000} = 2 \cdot 8$$

$$\text{,, after ,, ,,} = \frac{42{,}482 \cdot 92 \times 1 \cdot 537}{33{,}000} = 1 \cdot 97.$$

Example.—If the difference in temperature between the up-cast and down-cast shafts is increased four times, how is the pressure and quantity altered?

(1) The pressure varies as the difference of temperature between the up-cast and down-cast, so if the latter increases four times, the pressure will also be increased four times.

(2) The quantity varies as $\sqrt{\text{difference of temperature between the two shafts}}$. Therefore increasing the temperature four times increases the quantity by the $\sqrt{4} =$ twice.

CHAPTER XIV.

SAFETY LAMPS.

It is now over eighty years since Sir Humphry Davy crowned his long and patient researches by the invention of his safety lamp, an event which marked a new epoch in coal-mining. Before Davy's time the miner had to rely on very insecure methods for detecting or for working in the presence ~~of fire-damp~~. At the beginning of this century Spedding's steel mill was the only apparatus which enabled him to continue his work in the presence of small accumulations of gas. By this machine, which was soon demonstrated to be unsafe, and which was the cause of numberless explosions, the miner was enabled to work by the faint light of an intermittent spark resulting from the contact of a piece of flint with a revolving steel disc.

Since the introduction of the Davy lamp, large numbers of other safety lamps have been patented, but all are on practically the same principle as Sir Humphry Davy's, although some are, of course, a great improvement on the original, both as regards safety and lighting power.

Definition.—"Safety lamps are contrivances by which a light, surrounded by an explosive mixture of fire-damp and air, may be maintained in lamps without communicating flame to the outside atmosphere." As at present constructed, they depend upon the fact that flame, when brought in contact with wire gauze of certain degrees of fineness, cannot pass through it, owing to the rapidity with which the heat is conducted away, so that it cannot be communicated to the outside atmosphere.

Sir Humphry Davy first demonstrated this by some experiments he made with metallic tubes. He found that it was easy enough to effect an explosion of fire-damp and air in a wide vessel, but that it was impossible to effect it in a narrow metallic tube.

Metallic tubes of $\frac{1}{8}$ of an inch in diameter and $1\frac{1}{2}$ inches long prevented an explosion, and this phenomenon, according to Davy, probably depended "upon the heat lost during the explosion in contact with so great a cooling surface, which brings the temperature of the first

portions exploded below that required for the firing of the other portions; and it has been already shown that the fire-damp requires a very strong heat for its inflammation. Mixture of the gas with air I found, likewise, would not explode in metallic canals or troughs, when their diameter was $\frac{1}{7}$ of an inch., and their depth considerable in proportion to their diameter, nor could explosions be made to pass through such canals. Explosions, likewise, I found would not pass through very fine wire sieves or wire gauze." Now wire gauze is nothing more than a series of small tubes, having very small diameters, and of very short lengths. The wire gauze mostly used for safety lamps has 784 apertures to the sq. in.

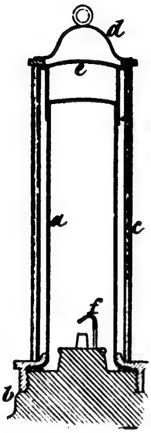


FIG. 444.—Davy Lamp.

Davy Lamp.—This lamp, as originally constructed, consisted of a small cylindrical vessel *b* for holding the wick and oil, provided at the bottom with a pricker *f* for trimming the former, and surmounted by a cylinder of wire gauze *a*, made double at the top, and supported by small iron rods *c*, terminating in the cover *d*, to which is attached a handle. The gauze cylinder is about $1\frac{1}{2}$ in. diameter, and 7 in. long, with wires about $\frac{1}{50}$ of an inch in diameter, having, as already stated, about 784 openings per sq. in., through which air enters to keep the flame burning freely. If a certain percentage of fire-damp enters along with the air, the mixture will ignite and fill the space inside the gauze with flame; but as soon as this flame comes into contact with the wire gauze, it is immediately cooled down, and cannot pass through the opening unless it is allowed to burn until the gauze becomes heated to a certain temperature, when the flame can pass through to the outside.

An explosion may be brought about by a 'Davy' lamp in several ways, such as:—

By allowing gas to burn inside until the gauze becomes red hot.
By allowing a strong current of air to blow against the lamp, thus forcing the flame through the gauze, which occurs when the air attains a velocity of about 5 ft. per second.
By a sudden jerk or shock to the lamp, or by a shock due to heavy blasting operations.
By the miner carelessly damaging his lamp or opening it in the presence of an explosive mixture.

The great disadvantage of the 'Davy' lamp is the very poor light that it gives, $\frac{1}{15}$ to $\frac{1}{30}$ of a candle power only.

The 'Davy' lamp in its original form is little used now, being unsafe in most collieries, where the air now travels at such high velocities round the workings.

Clanny Lamp.—This lamp is similar to the 'Davy' lamp in con-

struction, but has a glass cylinder *a* instead of the lower portion of the gauze, which enables it to give a much better light, and to be more easily carried in an air current; but, on the other hand, it causes an explosion more readily, owing to the area of gauze cylinder being smaller than in a 'Davy.' It is also unsafe in a very strong current of air, as it will readily pass the flame when the air is travelling at a velocity of 6 or 7 ft. per second. This lamp is not now used unless it has an additional protection in the shape of an iron shield surrounding the outside of the gauze.

Stephenson Lamp.—The essential points of dissimilarity between a 'Davy' lamp and a 'Stephenson' lamp are that, whereas in the former the flame is simply surrounded by a wire gauze through

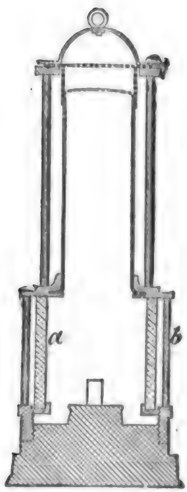


FIG. 445.—Clanny Lamp.

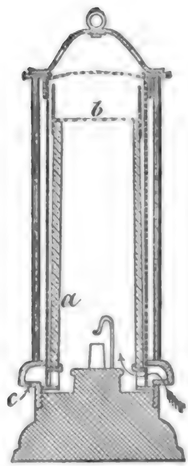


FIG. 446.—Stephenson Lamp.

which the air passes, in the latter the flame is surrounded and burns within a glass cylinder *a*, covered at the top with a perforated copper cap *b*, and is fed by air passing through perforations in a metal ring *c* at the bottom. The lamp has in addition the wire gauze of the 'Davy,' and if the glass happens to get broken it still remains safe.

Like the 'Davy,' the 'Stephenson' lamp has a very small illuminating power, and is very readily extinguished, but is very much safer than the former in air currents, as it will not 'pass' the flame until the velocity of the air reaches 8 to 10 ft. per second.

This lamp is now little, if at all, used in fiery mines.

Marsaut Lamp.—The 'Marsaut' lamp, which was the invention of a well-known French mining engineer, differs very little from the 'Clanny' lamp. Instead of having a single gauze like the

'Clanny,' it is provided with two or three gauzes fitted into the inside of each other, which tends greatly to increase the safety of the lamp.

As used in Britain it is made with two gauzes only, with the addition of an iron shield as a further protection, and in this form it is a very safe kind of lamp for use in fiery mines.

The Marsaut lamp is largely used both in England and Scotland, and has much to recommend it, as it does not pass the flame until a very high velocity of air current is reached, and it has the further advantage of not being so easily extinguished as some other forms of safety lamps. For oncast men especially, such as roadsmen, pony-drivers, etc., the Marsaut is to be recommended, as the nature of work those persons are engaged in varies greatly from that of

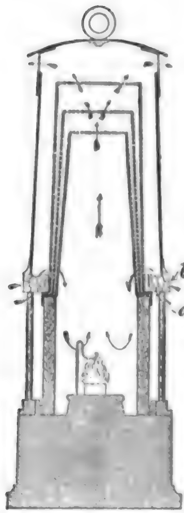


Fig. 447.—Marsaut Lamp.

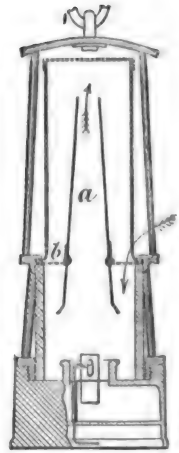


Fig. 448.—Mueseler Lamp.

a miner working at the coal-face, where a lamp is used under more favourable circumstances, being in most cases kept perfectly still or at most moved about only over small areas. In collieries where there is only a small amount of fire-damp given off, this lamp is often used without the outer shield, and with only two gauzes, which makes it much better suited for underground requirements, as the amount of light given out is much increased by the freer supply of air that the flame receives, and the latter is not so easily extinguished as when the shield is used.

Mueseler Lamp.—The Mueseler lamp is constructed somewhat like the Marsaut. It has the glass cylinder round the flame in the same way, but instead of having two or three gauzes it has a single gauze only, with a conical-shaped metal chimney *a* (fig. 448)

fitted inside, immediately above the glass cylinder, to which it is fixed by a ring of gauze *b*. The air to feed the flame passes first through the outer gauze, then through the gauze cap between the metal chimney and the glass to the flame, and the products of combustion pass up the metal chimney through the gauze cylinder and into the atmosphere. The metal chimney has thus a double purpose to serve, viz., to create a strong upward draught and to insure the inlet air being drawn down close to the glass to keep it cool. When fire-damp is suddenly ignited in this lamp the resulting gases, principally CO_2, fill up the conical chimney and speedily extinguish the flame.

The great drawback to this lamp is the readiness with which the light is extinguished, it being very sensitive to the least shock or jerk, while, if held slightly out of the perpendicular, the light at once goes out through the supply of air being cut off. Nevertheless, it is extensively used both in Britain and in other European countries, and is both good and safe where there is plenty of ventilation.

The Royal Commissioners on Accidents in Mines objected to this lamp on the following grounds:—

(1) It is very easily put out.

(2) The glass is easily broken by a blow, by the flame playing on it, or by cold water coming in contact with the hot glass.

(3) There are difficulties in getting tight joints where the metal ring and glass cylinder meet.

(4) Difficulties arise from combustion with a tendency to smoke the glass, thus lowering the illuminating power.

These objections might also be urged against nearly all the safety lamps at present in use.

Hepplewhite-Gray Lamp.—In this lamp and its newer modifications, the construction differs somewhat from any other safety lamp; the differences consisting chiefly of the manner the air is admitted to feed the flame, and also in the shape of the glass cylinder surrounding the latter (fig. 449).

The standards for supporting the lamp, instead of being solid, as they are in other lamps, are made of tubes down which the air passes to an annular chamber, situated immediately over the oil vessel, and protected by wire gauze.

In the form in which this lamp is now made, there are only three inlet tubes instead of four as formerly, one of the tubes being considerably broader than the others and acting as a deflector.

The glass surrounding the flame, instead of being cylindrical in shape, is made in the form of a cone, and is very much longer than the glass of an ordinary safety lamp. Immediately above the glass is a gauze, also of conical shape, and outside that a cone of metal, the whole forming a very strong compact lamp. In addition it is fitted with a shut-off arrangement, so that the air can be admitted either at the top or bottom of the tubes. In the new form a shield-plate *a a*, which is part of the hood, projects over and completely covers

the top of the inlet tubes, and prevents the lamp from being extinguished as readily as the older type. The air passes down the tubes, while the products of combustion pass upwards and out into the atmosphere by two horizontal rows of holes *b b*, in the hood *c*. To facilitate the cleaning of the lamp, the ring securing the glass is screwed on to a vertical plate *d*, which forms the air inlet. This arrangement enables the whole of the inside to be quickly and easily removed when the lower gauze ring is unscrewed. It is claimed for this lamp that it gives a much superior light to any other form, owing to the shape of the glass, which allows the rays of light to be projected in all directions, thus permitting the roof to be examined with ease without tilting the lamp. It is also claimed for it that very small quantities of gas can be readily detected by its means. The author has had little practical experience of this lamp, but some time ago some practical firemen and overmen, who were attending a class in Lanarkshire to which he was lecturing, mostly made, under his directions, a number of tests with the Gray lamp, and at the same time with Mueseler and Marsaut lamps; and reported that the former could be used to detect small quantities of gas much more easily and quickly than either of the others. The only objections to the use of the Gray lamp are its weight and expensiveness, but there can be little doubt that it is a very superior lamp for use underground. It is very safe in every respect and can withstand almost any current of air met with in collieries, having been tested with velocities up to 100 ft. per second without passing the flame.

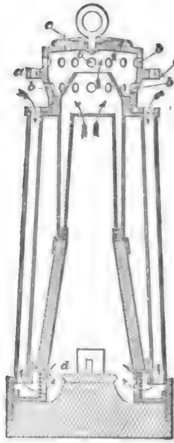


Fig. 449.—Hepplewhite-Gray Lamp.

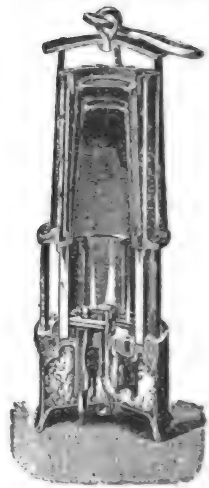


Fig 450.—Wolf Lamp.

Wolf Safety Lamp.—This safety lamp, constructed to burn benzine, is very largely used on the Continent. It is somewhat similar to an ordinary Marsaut lamp, with two gauzes, and is provided with a corrugated shield, provided with apertures, which is said to give it additional safety and allows of its burning more freely.

It is also provided with an 'igniter' for relighting it if extin-

guished, without rendering it necessary to unscrew the bottom of the lamp, which is a great advantage. The fastening is secured by a magnetic lock, and cannot be opened without the aid of a powerful magnet. It is said to give a much better light than an ordinary safety lamp burning oil, and can withstand strong air currents, and detect small percentages of gas, while the cost of fuel is low compared with other lamps.

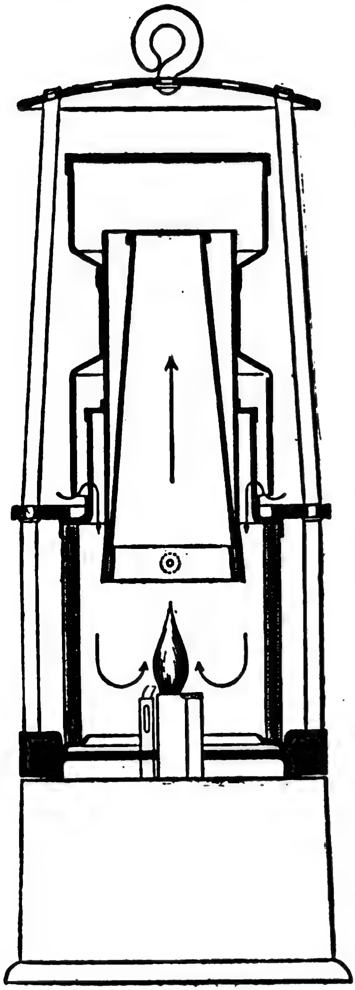


FIG. 451.—Wolf-Dahlmann Lamp.

Wolf-Dahlmann Lamp.—This may be said to be an improved Wolf lamp. In it the air is conducted from above the flame through a gauze ring $1\frac{1}{2}$ ins. in height, which is covered by a movable brass cylinder. The products of combustion are carried off by an inner brass cylinder, at the top of which is fixed a wider cylinder (see fig. 445), the latter being capped with wire gauze. Inside the inner brass cylinder is a gauze chimney, so arranged that it can be taken out for the purposes of cleaning. By means of this simple arrangement the products of combustion rapidly pass off, while fresh air enters from all sides through the gauze to the flame without becoming mixed with the products of combustion. In this manner a good circulation results, which causes the flame to burn steadily and brightly, while it is easy to light the lamp while locked by means of the igniter which is placed within it.

* *Evan Evan's Lamp.*—This is a bonneted Clanny lamp, with the bonnet, which extends from the flange above the glass to the dome of the lamp case, fixed permanently. The air is admitted through a series of holes in the horizontal flange above the glass. The products

* *Report of Royal Commission on Accidents in Mines*, p. 206.

of combustion escape through a series of holes in the top of the bonnet (see fig. 452). This lamp is also provided with an automatic arrangement for closing it in the event of gas becoming ignited inside the gauze, the main features of this arrangement being as follows:—Within the bonnet and surrounding the gauze are two cylinders of the same height closely fitting one another. The inner cylinder is open at the top and is perforated near the bottom with fifteen holes, each $\frac{1}{4}$ in. in diameter. The outer cylinder, which is closed at the top, is perforated near the top, with a similar series of holes. A rod slides through a tube and is maintained in position by a loop of thread close to the gauze and stretched between two hooks. If this loop gets burned through, by gas burning inside, the rod is

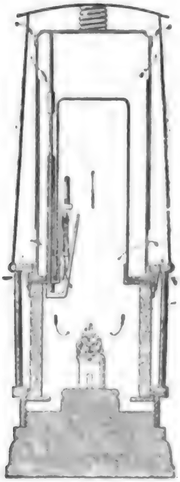


FIG. 452.—Evan Evan's Lamp.

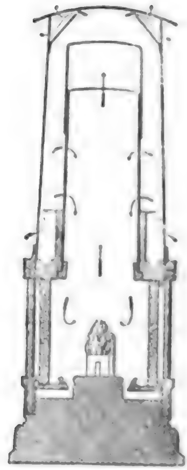


Fig. 453.—Evan Thomas Lamp.

no longer held in position, and a strong spiral spring between the top of the outer cylinder and the dome of the lamp pushes the former down over the inner cylinder, and thus closes both series of holes simultaneously, and extinguishes the flame in a few seconds.

Evan Thomas Lamp.—This was one of the lamps which was tested by the Mines Commission, and reported upon as giving very satisfactory results. In its principles and construction it is an improved form of the Clanny lamp. At the bottom of the gauze cylinder is a close fitting brass ring or tube about 1 in. high, the top of the ring terminating in a horizontal flange, which extends to within about $\frac{1}{16}$ of an inch of the outside shield or bonnet. Through this small space the inlet air is admitted to feed the flame, the products of combustion passing out at the top of the bonnet.

The Evan Thomas lamp is simple in construction, gives a fairly good light, and is safe in currents of air travelling with a velocity of over 50 ft. per second. It has also the advantage of not being very readily extinguished by a sudden jerk or by being held on the slant. In an explosive atmosphere, however, it soon becomes extinguished.

The following table of experiments with safety lamps in currents of air of different velocities is interesting, as it shows the behaviour of different lamps and the ratio of safety under these conditions.

Name of Lamp.	Percentage of Gas present in air.	Velocity of Air in ft. per minute.	Duration of Experiment.	Results.
Davy lamp, . .	7 per cent.	600	10 seconds.	Explosion.
,, . .	7 ,,	400	60 ,,	,,
,, . .	7 ,,	370	180 ,,	No explosion.
Davy (shielded), .	7 ,,	800	60 ,,	,,
,, (in can), .	7 ,,	1500	60 ,,	,,
Stephenson lamp safe up to 800 ft. per minute.				
Clanny ,, ,, 600 ,, ,,				
Mueseler lamp, .	8 per cent.	2888	5 ,,	Lamp went out.
,, ,, .	13 ,,	2888	75 ,,	Continued to burn.
,, ,, .	13 ,,	600	6 ,,	Explosion.

The Marsaut lamp gave the same results as the Mueseler, but it was found that if the current of air was reversed while being admitted to the flame, an explosion occurred in 5 seconds. The number of safety lamps that have been patented and placed on the market is so large that it would take a large volume to fully describe them all. One writer enumerates over seventy, and that by no means exhausts the list.

Fire-damp Indicators.—Within the last few years a large number of instruments for detecting small percentages of fire-damp have been brought before the mining public, but very few of them are of much practical use to the miner, most of them being too complicated in design, too sensitive to be handled freely, and too expensive.

Hydrogen Indicator.—It has long been known that the pale hot flame of hydrogen gas is very sensitive to the presence of fire-damp, even when the latter is present in very small quantities. Professor Clowes has invented a lamp in which a hydrogen flame is used for the detection of small percentages of CH_4. The accompanying figure (454) shows this apparatus as used in conjunction with an ordinary Heppelwhite-Gray safety lamp. A small cylinder C, containing hydrogen at very high pressure, is attached to the lamp, and at the bottom of the cylinder a tube B, of small diameter, is connected with the interior of the safety lamp, the top of the tube being just about on a level with the burning wick W. When a test is to be made, hydrogen is admitted through the small tube B, by opening a tap T, with a key. The flame of the ordinary wick immediately lights the hydrogen, the jet of which can be regulated to any required size of flame. The ordinary wick is then drawn down by

means of the pricker, and the large flame being extinguished, the indicator is ready to make a test with the hydrogen flame alone. A small ladder-like scale S is fixed inside beside the flame, to measure the percentage of gas found, each of the steps on the scale being a definite value. The lamp is guaranteed to measure as little as 0·25 per cent. of fire-damp in air. The small hydrogen cylinder can be detached and carried in the pocket when not required for testing, and the safety lamp can then be used in the ordinary way.

Stokes' Indicator.—In this lamp 'absolute alcohol' is employed to produce a flame for detecting the presence of small quantities of fire-damp. The indicator may be used with any ordinary safety lamp, and is a very simple arrangement, shown in fig. 455.

A small vessel *e e*, having a thin tube and wick, screws into an opening at the bottom of the safety lamp, the top of this tube reaching

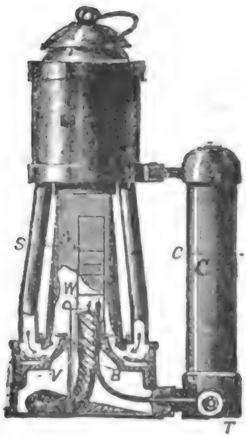


FIG. 454.—Hydrogen Indicator.

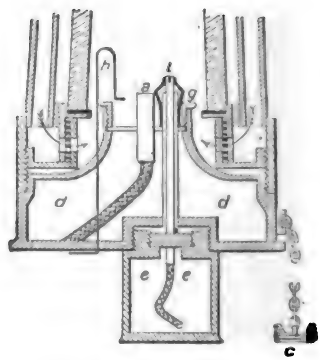


FIG. 455.—Stokes' Indicator.

to the top of the wick *a*, where it can be lighted; a slit *i*, at the top of the alcohol tube gives the standard flame for testing. When a test is about to be made, the brass plug *c*, which fits the opening at the bottom of the lamp, is unscrewed, and the tube of the indicator inserted. In a few seconds the heat will cause the alcohol to ascend and ignite at the oil flame. The oil wick is then drawn down by the pricker *h*, and the test proceeded with. After it is completed the oil wick can be raised and re-lighted at the alcohol flame. The alcohol vessel is then unscrewed, the plug *c* put in, and the lamp again becomes an ordinary safety lamp. It is said to be capable of indicating as little as ½ per cent. of CH_4 in air.

Pieler Lamp.—The Pieler indicator may be described as a large Davy lamp constructed to burn pure alcohol with a special wick.

The air which supplies the flame is admitted by a tube, protected by superposed discs of gauze, which pass vertically through the vessel containing the alcohol. Immediately above the burner and surrounding the flame, is a short, conical chimney, open at both ends, and before making tests, the flame should be adjusted in pure air, so that it comes exactly to the top of the chimney. When burning in a mixture of fire-damp and air the flame shows a much more conspicuous cap or 'halo' than can be produced by the flame of an ordinary safety lamp burning mineral or vegetable oil. A scale is fixed in front of the lamp for measuring the different percentages according to the height of the flame. The apparatus is said to be

FIG. 456.—Pieler Lamp.

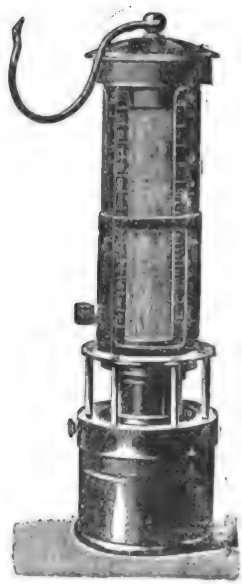


FIG. 457.—Chesneau Lamp.

capable of detecting the presence of ½ per cent. of fire-damp in air. While there is no doubt that it is a most sensitive gas-detector, it is in its present form practically useless for the ordinary usage to which safety lamps are subjected when testing for gas underground. It is too sensitive, too easily extinguished, and requires to be very carefully handled, as the vapour given off from the burning liquid is itself highly explosive when mixed with air. The lamp as now constructed is fitted with a shield and is much safer, in currents of air, than the original type.

Chesneau Lamp.—This apparatus is also constructed to burn

alcohol, and is somewhat like the Pieler indicator. It is constructed of a brass reservoir for the alcohol, surmounted by a circular crown for the admission of air (which can be regulated) through double gauzes. Resting on the crown and surrounding the wick-tube is a solid cylinder of sheet-iron, which serves as a screen. Above this screen and resting on it, is an iron wire gauze 5½ in. high. The gauze is surrounded by a sheet-iron shield furnished with an observation window of mica, on which is engraved a scale for measuring the percentages of fire-damp. The shield is fitted at its base with an annular diaphragm, which closely surrounds the base of the gauze, so that the exterior air does not impinge directly on the gauze.

The interior of the reservoir contains a small piece of cotton wool under the wick-tube, to prevent the rapid escape of alcohol if the lamp is overturned, while if the apparatus is laid horizontally it is at once extinguished. It is said that caps can be observed when as little as 0·1 to 0·2 per cent. of fire-damp is present in the air, and that the cap becomes quite marked when the percentages reach 0·5.

Like the 'Pieler' lamp it is best suited for making observations in a still atmosphere, which is not the ordinary condition of underground workings. In the 'Chesneau' lamp, dew forms on the sheet of mica, by the condensation of the aqueous vapour resulting from the condensation of the alcohol, aided by the cold external air impinging against it, and this prevents the observer from making accurate observations.

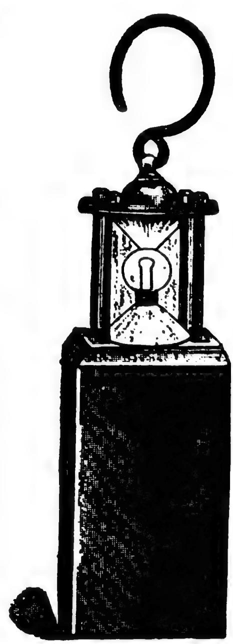


FIG. 458.—Sussman Electric Lamp.

Electric Lamps.—With the advent of electricity for illuminating purposes, much was hoped for in the way of lighting underground workings, but so far, with the exception of its use for lighting pit-bottoms and main roads, little progress has been made, all the electric safety lamps that have been invented up to the present time being of little use so far as practical work is concerned. Most of the electric lamps which have been introduced have been too large and unwieldy, burned for too short periods, were liable to go suddenly out, and were too expensive for colliery work. An even greater bar to their usefulness was that they were nearly all constructed on the 'wet battery' principle, which is not suitable for underground work, where lamps are often pretty roughly handled.

The nearest approach to a good portable safety lamp is that known

as the 'Sussman' patent electric lamp. The lamp is about the same size and weight as an ordinary safety lamp, measuring 2½ in. square and 8 in. high, and weighs about 3½ lbs.

It is constructed with a dry battery, which is a great advantage, as it can be handled much more freely without injury. It is also said to be impossible to ignite gas with it, even although the lamp gets broken. One defect of these electric lamps is that they give no indication of the presence of gas, which is a serious drawback, especially in fiery collieries.

Construction of Safety Lamps.—In all the improvements and modifications that have been made in safety lamps of late years, the tendency has been in one direction, viz., to render them as safe as possible in currents of air travelling at high velocities. While this object is one to be commended, a great deal more than this is required of a good safety lamp, and if some of the inventors would turn their attention to combining safety with good illuminating power and simplicity of construction, they would render good service to those who are compelled to use such lamps. A very large number of safety lamps, which give a good light when on the surface, or in the main airways underground, are of little or no use under the ordinary conditions met with in mines, either owing to the small amount of light which they give, or the sensitiveness which shows itself by the ease with which the light becomes extinguished. Until a safety lamp that will give a light equal to at least one candle power can be placed in the miner's hands, it cannot be said that the limit of improvement has been reached.

Lighting Power of Lamps.—The lighting power of lamps is a very variable quantity, and differs very much under different conditions. To obtain good results there must be a good burning oil, a fairly large wick and burner, and a glass so made that it will diffuse the light in every direction. With safety lamps, under ordinary conditions, it takes about three Mueseler's or Marsaut's to yield one candle power.

Some tests were made a few years ago with various safety lamps at the Hamilton Gasworks, and the following results were obtained :—

Name of Lamp.	Number of Lamps equal to one Standard Candle.	Oil burned.
Naked light, ...	0·76	Paraffin wax.
,, ...	0·60	Sweet oil.
Davy lamp, ...	6·17	,,
Mueseler (Belgian), . . .	2·51	,,
,, (protector), . .	2·02	Naphtha spirits.
Williamson lamp, . . .	1·54	Sweet oil.

* The following table is given by Professor Lupton, showing the

* Lupton's *Mining*, p. 288.

light of different safety lamps compared with that of a standard candle:

Name of Safety Lamp.	Number of lamps required to equal one standard sperm candle, the illuminating object being on a level with the flame.	Number of lamps required to equal one sperm candle, allowing for the shade cast by lamp cover, bottom, and pillar, the candle and the lamp being each in a cylinder of white tracing paper 2 ft. high and 8 in. diameter.
Davy lamp, good flame, . .	13 (average)	26 (average)
Protector shielded Marsaut, .	3 ,,	...
,, ,, flame average height, . .	$4\frac{1}{4}$,,	...
Deflector lamp, large flat flame,	3 ,,	...
,, ,, moderate clear,	5 ,,	...
Clifford lamp, very good flame,	$1\frac{9}{10}$,,	...
,, ,, moderate flame,	$2\frac{1}{2}$,,	5 (average)
Ashworth lamp, . . .	$3\frac{1}{2}$,,	$11\frac{1}{2}$,,
Stephenson lamp, good flame,	$7\frac{1}{2}$,,	23 ,,
Tallow candle, good average flame, ...	$1\frac{3}{4}$,,	...

The light of safety lamps will depend a good deal on the sort of oil used and the state of the ventilation; light oils, such as petroleum and colzaline, giving a much better and clearer light than heavy oils, such as rape or seal oil.

If light oils are used, they require to be very carefully handled, and no naked lights brought near the oil receiver, as they give off inflammable gases at comparatively low temperatures, and are apt to catch fire very readily and do much damage. The writer has known at least half-a-dozen cases where lamp rooms have been set on fire and destroyed through careless handling of these light oils.

The Royal Commission on Accidents in Mines recommended a mixture of vegetable or animal oil with about one-half its volume of a petroleum oil of safe flashing point, as giving the best results as an illuminant for safety lamps.

Cost of Upkeep of Lamps.—The average cost of safety lamps—if used with light oils, which are cheaper than the heavy vegetable oils,—including both oil and wick, is about a halfpenny per lamp per shift. Thus Davy lamps cost about 0·375d. per day for oil alone; Mueseler lamps about 0·295d.; while the cost of naked lights may be estimated at 1·25d. per diem.

The following is a detailed statement of the cost of safety lamps per annum, as ascertained at the Clyde Collieries, Hamilton, although the writer thinks that some of the items, such as gauzes and glasses, are underestimated:—

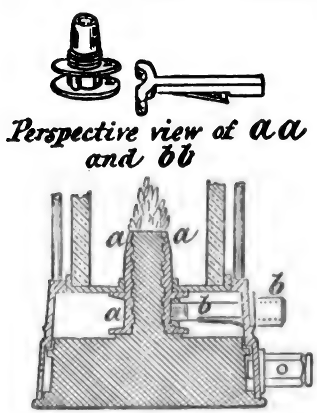
	s.	d.
Cost of oil (300 days) per lamp, ...	7	$4\frac{1}{2}$
Repairs, ,, ,, ...	0	$4\frac{1}{2}$
Gauzes and glasses (300 days) per lamp, ...	0	$2\frac{3}{4}$
Lampmen's wages, ,, ,, ...	9	$3\frac{1}{2}$
Total cost, . .	17	$8\frac{1}{4}$

Locking Contrivances.—Nothing conduces more to the safety of a lamp, or of those using it, than an efficient method of locking; for if the cause of many explosions could be accurately ascertained, it would be found that not a few of them were caused by miners themselves surreptitiously opening their lamps; indeed, in many explosions, this has been clearly ascertained to have caused the disaster. It is a notorious fact that few men are more careless of their own safety than miners, who have been known to open their lamps at the face, even in the neighbourhood of large quantities of fire-damp.

In the old method of locking lamps, a padlock was often used, and could be opened easily by a duplicate or skeleton key. Another lock which was largely used, and is still used to some extent, was a small screw bolt, either with a square or tapered head, which was turned until the body of the lamp and the bottom were fastened together, by means of the bolt pinching the bottom part. This lock was of little use, as any workman with an old nail filed to the proper size could open his lamp by unscrewing the lock. The simplest and best method of locking a lamp is by using a riveted lead plug, connecting the body of the lamp with the oil-vessel. The lead plug should be firmly riveted, and each end stamped with letters or marks, varied from day to day.

Without this precaution it has been found that the rivets can be removed and replaced in such a way as to render detection extremely difficult.

Magnetic locks are also used, in which the lamp can only be opened by the aid of a powerful magnet. Of this class, Wolf's magnetic lock is the simplest and most satisfactory.



FIGS. 459, 460.—'Protector' Lock.

Protector Lock.—Many lamps are now fitted with the 'Protector' arrangement for securing the oil-vessel to the top part of the lamp. In addition to this arrangement the lamp is also locked in the usual way.

The apparatus will be understood from figs. 459, 460. The wick-tube has a screw thread upon it throughout its whole length, and on this is screwed a 'thimble' *a a*, provided at its lower end with a flange, on the outer end of which a screw-thread is also cut. By the latter screw the oil-cup is attached to the lamp, and, when this has been done, the flanged thimble is fastened in position by a bolt *b*, provided with a spring; the bolt cannot, therefore, be withdrawn until the oil-cup is removed. The thimble is screwed on to the wick-tube

before the wick is lighted. The oil-vessel can, of course, be readily removed, but only by making the end of the wick-tube traverse the closely-fitting thimble above mentioned for a distance of 1 to 1·5 in., and during this process the diminished flame of the naphtha spirit is certain to be extinguished. With this arrangement it is practically impossible to open the lamp without extinguishing the flame, unless the lock bolt gets broken or the spring attached gets out of order, which does not readily happen with fair treatment.

As already stated, the oil-vessel is fastened in the ordinary method by riveting with a lead plug, which gives additional security, and also permits the oil-vessel to be partially unscrewed for the purpose of regulating the flame, but prevents it from being altogether withdrawn.

Testing Lamps.—All safety lamps, before being taken into the mine, require to be tested at the lamp station on the surface, or at a station at the bottom of the shaft, to see that they are in a safe condition for using, and also to ascertain that the parts are properly fitted together, particularly the glass and the connecting parts at the top and bottom of it, for it is absolutely essential to safety that no opening should be left at the junction of the glass with the brass rings for an explosive mixture to enter the lamp and ignite at the flame.

The testing of the lamps is often done in a very cursory way at many collieries, owing, no doubt, to the lamp attendant not being aware of the true value of such testing.

The simplest method of testing safety lamps is to take a brass tube of small diameter, somewhat like a blow-pipe, and to blow through it with the mouth so that a current of air impinges against the glass all round the top and bottom edges. If the glass is not properly fitted, it will be detected by the current of air getting in at the edges and deflecting the flame or extinguishing it altogether.

Sometimes the lamps are tested in a more scientific manner by being inserted in an inflammable mixture to see if they are really 'safe,' this method being much better than the blow-pipe test.

* A testing apparatus for safety lamps is shown in fig. 461.

A and B are two cylinders which fit into one another. The cylinder A is filled with water to about three-fourths of its capacity, and the cylinder B, the bottom of which is open, contains a valve at the top of it. By simple pressure of the thumb, the valve can be opened, whereupon cylinder B is made to rise to three-fourths of its height, and become filled with air. At the bottom of the cylinder A is a pipe, which, as it extends above the surface of the water, receives the air that cylinder B forces into the reservoir C, where the gases are generated. C is a reservoir, the inside of which is divided by several partitions, made of corrugated iron, and filled with some absorbent material such as cotton wool. The glass cylinder D is filled with benzine immediately before the apparatus is required for

* The author is indebted to the makers, Messrs Friemann & Wolf, for the diagram and description of this apparatus.

use. If the cock E be opened, the benzine will flow into the reservoir C, in which it is obliged to follow the windings of the sheet iron, and in this manner offers a large surface to the air, the latter being thus impregnated with benzine and transformed into gas.

The gas then passes through the cock F into the testing cylinder G, which is made of tin-plate, and fitted with a pane of glass. Inside this testing cylinder there is a spiral pipe, the inside of which is drilled with small holes which serve for the admission of air. By means of a metal sliding valve the quantity of air entering may be regulated. The lighted lamp is placed within the spiral pipe in the

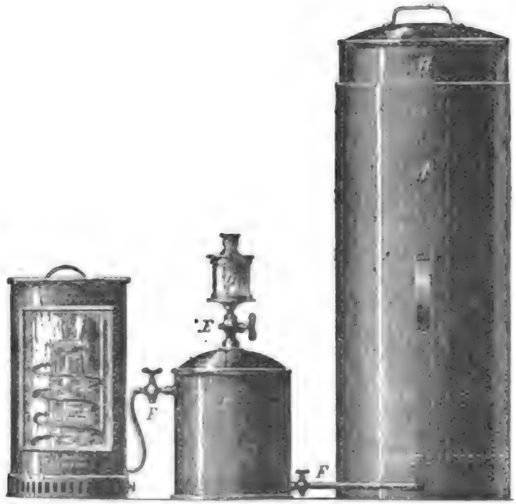


FIG. 461.—Testing Apparatus for Safety Lamps.

cylinder G, and the cock F is opened. The gas issues through the fine holes in the pipe and streams on to the lamp.

If the latter be defective, the generated gas will be ignited in the cylinder G, but the flame of a well-fitted lamp will be extinguished in the presence of too large a quantity of this gas, which will produce the same effect upon the flame as fire-damp does.

To make more elaborate tests of safety lamps in explosive gases, a larger and more costly apparatus may be employed. The apparatus will be understood from fig. 462. It consists of a tank A, surrounded by water like an ordinary gasholder. Connected to this tank are two pipes C and D, of 2 in. diameter. The pipe C is connected to the mouth of a long wooden box B, about 26 ft. long, 10¼ in. high, and 4¾ in. wide, constructed of boards 1¼ in. thick, carefully

jointed together, with all cracks or seams completely closed. On the top of this box, and at distances of 2½ ft. apart, are openings which are closed with covers H H, which are easily displaced by any explosion of gas which may occur in the box, and permit the exploded gas to escape.

To produce changes in the direction of the current of mixed gases from the horizontal, upward or downward, according as it may be desired that the gas should impinge upon the burning lamp, arrangements have been made to adjust the lamps inside to different levels (see fig. 462).

In order to make the necessary observations on the conduct of the lamps during the test, a part of the wooden box is inclosed by a wooden partition I, in such a way as to entirely darken it, and enable the observation of every change in the flame of the safety lamp to be made. The somewhat dangerous observation of the safety lamp

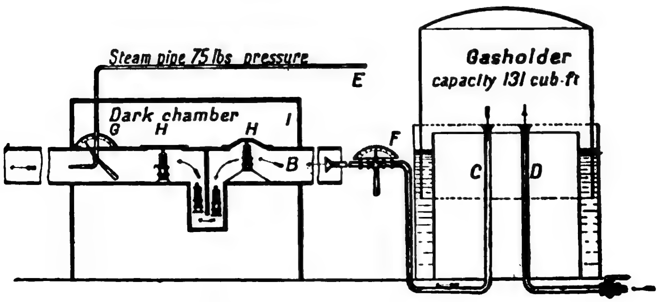


FIG. 462.—Special Testing Apparatus.

during a test is made through a glass plate ⅓ in. thick, which is set in the door through which the lamp is placed in the box.

The velocity of the air current or gas mixture, which can be increased up to 59 ft. per second, is produced by means of a steam nozzle E, the steam having a pressure of 75 lbs. per sq. in. enters the nozzle through a valve to which an arm K is attached, by means of which it can be adjusted so as to produce the desired velocity. In order to adjust and indicate the velocity an anemometer is used, and a scale constructed which indicates the velocity by means of a pointer attached to the arm.

The gas is conducted to the apparatus by the pipe D into the gas tank, to which is attached a meter for measuring accurately the quantity of gas entering the holder.

By weighting the gas tank, the gas will be forced down the pipe C into the wooden box, the inflow being regulated and adjusted by a nozzle having a pointer and scale F attached, to show the *percentage* of the gas mixture.

The end of the pipe C, where it enters the box, terminates in a funnel-shaped opening which is covered with a gauze so as to prevent the flame, in case of an explosion caused by a lamp, from entering the gas pipe and communicating with the gas-holder.

This apparatus can only be used when a supply of coal gas is available.

Trimming and Cleaning Lamps.—Probably nothing is more important as regards safety lamps than proper cleaning, trimming and fitting together of the various parts of which they are composed, for this part of the work bears directly upon the safety of the whole colliery.

While the object to be aimed at is to get a good safe lamp, giving out a good light, simplicity of construction is also to be greatly desired, and this is wanting in many lamps brought before the public. When it is considered that at some collieries from 500 to 1000 safety lamps have to be cleaned, trimmed, and fitted together every shift, the necessity of simplicity in construction is at once apparent, because, if a lamp is complicated and consists of a great many parts, there is a great probability that, in preparing a large number for use, some may be imperfectly put together, and however safe a lamp may be when in perfect order, it may become most unsafe under these conditions. The principal parts of a safety lamp which are liable to get out of order, or may be improperly fitted together, are the gauzes, the glass chimney, and the joints between the latter and the metal. The glasses used for safety lamps should be of the best quality, ground smooth, and parallel at the edges; glasses with chipped or rough edges should be discarded, for it is almost impossible to get tight joints where they come in contact with the metal rings. In the latter, 'washers' of asbestos millboard should be introduced, which will give a good bearing surface to the edges of the glass. Washers of leather or india-rubber are not to be recommended, as they are perishable and shift their positions if they become strongly heated, which often occurs. The gauzes should also be carefully examined daily, and thoroughly cleaned so as to free them from any oil or coal dust which may have clogged the meshes while underground. At the majority of collieries the gauzes are cleaned by hand, and where the lamps are not very dirty this does well enough, if good brushes of the proper size and quality are used, but in collieries where the workings are muddy and the dirt adheres firmly to the gauze, the lamps are more difficult to clean, as the dirt adheres strongly to the gauze and cannot easily be removed. When this is the case, the lighting power of the lamp will become very inefficient, as the necessary air for proper combustion is unable to reach the flame.

Machines either worked by hand or steam power are now used for cleaning the gauzes and glasses. By this means the gauzes are cleaned more thoroughly, and are not subjected to such rough treat-

ment as when cleaned by hand, there being also an economy in labour, an experienced man being able to clean 300 to 400 gauzes per hour.

Filling and Lighting Lamps.—As already stated, the utmost care should be exercised in charging safety lamps with oil, especially if the oil used is what is termed a 'light' oil, such as petroleum, naphtha or benzine, either of which is of a highly inflammable nature, especially when the temperature reaches a certain point. These oils, as a rule, have a very low 'flash' point (about 70° F.), and when they reach this point they give off a dangerously combustible vapour which will readily explode if brought in contact with a light. At collieries where these oils are used, the lamp-room should be so arranged that the lamps can be filled in a separate apartment to that in which they are lighted before distribution to the men. It will also increase the security against fire or explosion if the filling tank containing the oil is situated in a special vault outside the lamp-room, and a pipe led from it to where the lamps are filled.

Dripping pans should always be provided below each filling tap to receive any excess of oil. The dripping pans should be emptied at frequent intervals, as it is often from the overflowing or upsetting of one of these that accidents occur.

Tanks which work automatically are sometimes used for filling the lamps. Such an apparatus consists of an iron tank which holds 10 to 15 gallons of oil, and by means of a three-way cock placed at the bottom of the tank, the small quantity of naphtha necessary for the filling of a lamp is drawn from the tank into the glass reservoir, and by the turning of the cock is entirely shut off from that in the tank. When the sponge in the lamp will absorb no more oil, the outflow from the reservoir stops automatically. By this arrangement safety and economy are secured.

Note.—For students who desire more detailed information as to safety lamps, the author would recommend *The Report of the Royal Commission on Accidents in Mines*, 1886, where over one hundred such lamps are figured and described.

CHAPTER XV.

SURFACE ARRANGEMENTS, COAL CLEANING, ETC.

Siding Accommodation.—For the handling of a large daily output of coal, plenty of siding accommodation both for loaded and empty waggons should be provided. Many large collieries provide sidings for 200 or 300 loaded trucks, or for one day's output. This will be all the more necessary if the colliery is situated at a considerable distance from the main line, and a clearance is effected only once or twice daily. With the modern practice of separating the coal into many different classes for the market, the number of sidings requires to be greater than formerly, when it was customary to simply separate the dross from the round coal. From various causes the ground at

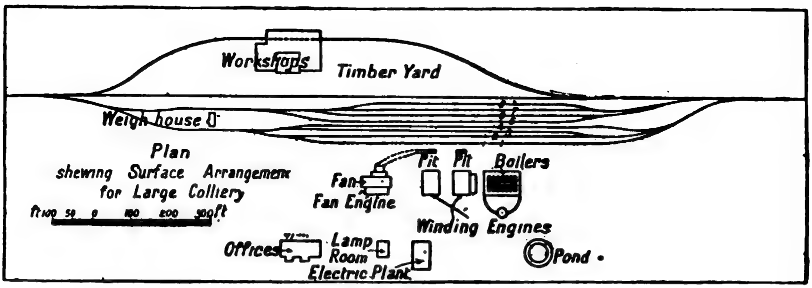


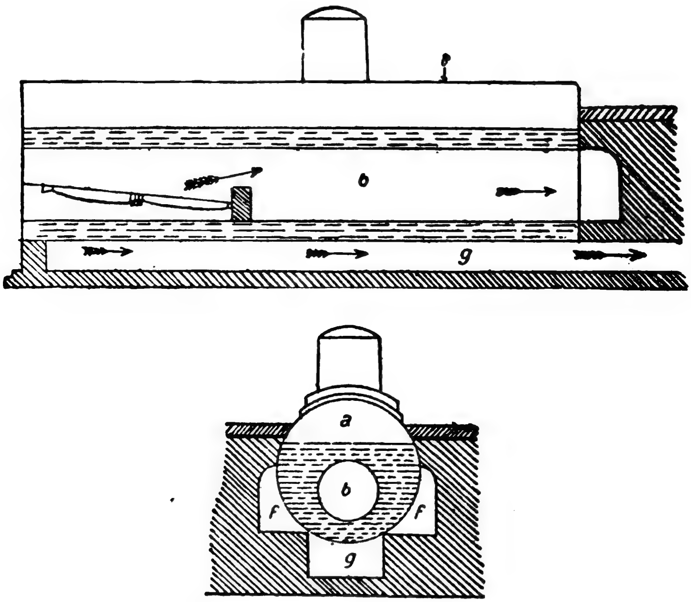
FIG. 463.—Plan of Surface Arrangements.

disposal may be limited in area for sidings, in which case it is impossible to provide large storage lyes for loaded waggons. As to the general arrangement of surface buildings, such as engine-houses, boilers, screens, etc., the area and disposition of the ground inclosed will largely determine the matter for each individual colliery. A general plan of surface arrangements is shown in fig. 463, taken from those of a large colliery raising from 1200 to 1500 tons of coal daily.

Inclination of Sidings.—The proper inclination for colliery sidings is an important matter. From the author's own experience he has

found that the best gradient behind the screens is about 1 in 70 for the empty waggons, with the inclination towards the shaft, while 1 in 60 to 66 is best suited for the loaded waggons in front of the screens. With these gradients the waggons should move freely in all weathers.

Boilers.—The boilers used at collieries may be of either the ordinary egg-end, or of the Lancashire or Cornish types. Where rapid steaming and high pressures are required, the Lancashire boiler is the best to use, and also, as a rule, more economical, convenient, and durable. If large grate area is required, low pressures, and low first cost, the egg-end type of boiler may be found better suited.



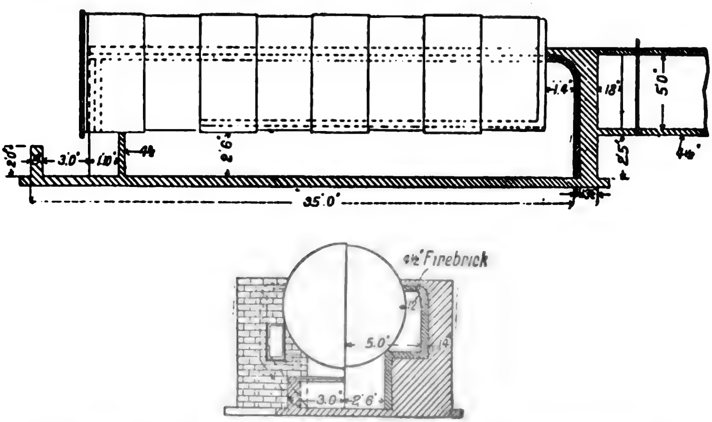
FIGS. 464, 465.—Longitudinal and Cross Sections of Cornish Boiler.

Cornish Boilers.—Cornish boilers consist of a cylindrical shell *a*, with flat ends, and having near the bottom a smaller shell or tube *b* (figs. 464, 465), which passes through the larger one and forms the furnace. The products of combustion pass from the furnace to the end of the tube, and return by the two side flues *f f* to the front of the boiler, pass into the bottom flue *g*, and so reach the chimney. By this arrangement the gases are reduced in temperature before coming into contact with the bottom of the boiler, where all the sediment collects, and there is therefore no danger of burning the plates on the under side of the boiler.

Lancashire Boilers.—This type of boiler differs from the Cornish in having two internal furnace tubes instead of one. The separate furnaces are intended to be fired alternately, so that while one is giving off smoke and unburnt gases, the other is burning briskly, and yielding its maximum heating effect. By this arrangement the mixture of smoke and unburnt gases from the 'green' fire are consumed in the flues, where they are raised to the necessary temperature by the gases coming from the bright fire.

The method of ventilation and draught is similar in the Lancashire to that in the Cornish boiler. The furnace gases pass to the end of the furnace tube, and thence by the flue underneath the boiler to the front, where they divide and pass by the side flues to the back of the boiler, from whence they escape into the chimney.

Figs. 465, 467 show a cross and longitudinal section of the seating of a Lancashire boiler. The side flues and closing tiles should be of the



FIGS. 465, 467.—Longitudinal and Cross Sections of Lancashire Boiler.

best firebrick. In order to increase the heating surface and promote a better circulation of water, the furnace tubes of Lancashire and Cornish boilers are often fitted with water tubes, those known as the 'Galloway' tubes being the best known and the most generally used. Such tubes have their disadvantages, however, as they tend to cool the furnace gases and retard combustion.

The Cornish boiler varies from 15 to 30 ft. in length by $5\frac{1}{2}$ to 8 ft. in diameter, the diameter of the furnace tube being $3\frac{3}{4}$ to $4\frac{1}{2}$ ft. The length of a Lancashire boiler varies from 20 to 35 ft., and the diameter from $5\frac{1}{2}$ to 8 ft., a size much used for colliery work being 27 ft. 6 in. × 7 ft. 6 in. diameter. The furnace tubes are usually about 2 ft. 6 in. or 2 ft. 9 in. diameter. Boilers of either type ought to be supplied with a safety valve, steam-gauge, glass water-

gauge, stop valve, water float, sludge, steam and feed water pipes, blow-off cock and pipe, damper, etc.

Accidents to Boilers.—Accidents to boilers are generally due to the following causes, viz., defective water supply, corrosion and incrustation of shell, or defective safety valves. Accidents also happen from plates getting worn thin, or rivets becoming fractured.

In the case of a defective water supply, if the water-level becomes dangerously low in the boiler the fire should on no account be withdrawn, as this procedure may accelerate an explosion by the sudden cooling of the plates; the fire should be damped with small coal or ashes, the damper put down, and the boiler allowed to cool gradually before water is put in.

Incrustation in steam boilers generally arises from the presence of acids in the feed water, such as sulphate or carbonate of lime. If the impurity is found to consist almost entirely of carbonate of lime, the feed water may be treated by the addition of caustic lime or milk of lime, or by what is known as Clark's process.

When carbonate and sulphate of lime are both known to be present, the feed water may be treated with caustic soda or soda ash. With both of these methods, considerable expense is involved, as large tanks are required to hold six to eight hours' water supply; and instead of treating the feed water in any way, it is allowed to enter the boiler and the lime precipitated in the boiler itself, and the sediment blown off frequently.

Boilers ought to be carefully inspected, internally and externally, and reported on every three months by a competent boiler inspector.

Evaporative Power of Boilers.—Theoretically 1 lb. of coal should evaporate from 12 to 16 lbs. of water, at 212° F., the actual amount depending on the quality of the coal. In practice a Lancashire boiler will evaporate 8 to 10½ lbs. of water at 212° F. per lb. of coal burned, and a Cornish boiler 7 to 9 lbs. of water per lb. of coal.

The weight of water which can be evaporated per hour for any given size of engine may be found by the following formula:

$$W = \frac{D^2 \times \cdot 7854 \times L \times 2 \times R \times 60 \times a}{1728}.$$

Where W equals the weight of water to be evaporated per hour; D the diameter of cylinder in inches; L the length of stroke in inches; R the revolutions of crank per minute, and a the value of 1 cub. ft. of steam at given pressure.

Example.—In an engine with a cylinder 20 in. diameter, a 24 in. stroke, and making eighty revolutions per minute, what amount of water would require to be evaporated per hour if the steam pressure is 65 lbs. per sq. in.?

Here the value of $a = \cdot 1538$ lb. at 65 lbs. absolute pressure.

$$\therefore W = \frac{20^2 \times \cdot 7854 \times 24 \times 2 \times 80 \times 60 \times \cdot 1538}{1728} = 6442 \cdot 34 \text{ lbs.}$$

Rate of Combustion.—The rate of combustion of fuel in the furnaces of steam boilers is usually expressed in pounds of coal consumed per sq. ft. of grate surface per hour. In ordinary boilers the rate of combustion is from 15 to 20 lbs. of coal per sq. ft. of grate per hour.

Strength of Boilers.—The strength of boilers depends on their construction and upon the material of which they are made. Boilers are now generally made of best mild steel, with the exception of the manhole mouth-piece and longitudinal bolt-stays which are usually of wrought-iron.

The tenacity of wrought-iron is given at 50000 lbs. per sq. in., and that of steel for boiler plate 80000 to 90000 lbs. per sq. in., and a factor of safety of 6 to 8 is usually allowed.

Let t represent the required thickness of plate in in.; p the working pressure in lbs. per sq. in.; d the diameter of boiler in in.; and M the factor of safety.

$$\text{Then for wrought-iron boilers } t=\frac{pd}{50000}\times \text{M} \text{ or } p=\frac{t\times 50000}{d\times \text{M}} \text{ (single riveted).}$$

$$\text{And } t=\frac{pd}{60000}\times \text{M (double riveted).}$$

$$\text{For steel boilers } t=\frac{pd}{70000}\times \text{M} \text{ or } p=\frac{t\times 70000}{d\times \text{M}} \text{ (single riveted).}$$

$$\text{And } t=\frac{pd}{90000}\times \text{M (double riveted).}$$

Example.—What thickness of double riveted boiler plate would be required for a steel boiler $7\frac{1}{2}$ ft. diameter to work at a pressure of 80 lbs. per sq. in.?

$$\text{Here } t=\frac{80\times 7\cdot 5\times 12}{90000}\times 8=\frac{6\times 8}{75}=\cdot 64 \text{ in. or about } \tfrac{11}{16} \text{ of an in.}$$

Example.—What would be the bursting pressure of a single riveted steel boiler 7 ft. diameter, with plates $\frac{5}{8}$ of an in. thick?

$$\text{Here } p=\frac{t\times 70000}{d}=\frac{\frac{5}{8}\times 70000}{84}=520\cdot 8 \text{ lbs. per sq. in.}$$

Low and Bevis give the following formula for the safe working pressure:

$p=\dfrac{8000\times 2t}{d}$; where t = thickness in in., and d = diameter of shell in in.

For the strength of boiler tubes, plain iron, Fairbairn gives the formula:

$\text{P}=\dfrac{806300\times t^{2\cdot 19}}{\text{L}\times \text{D}}$; where P = collapsing pressure per sq. in.; L = length of tube in ft., and D = diameter of tube in in.

Where great accuracy is not required t^2 may be substituted for $t^{2\cdot 19}$.

Number of Boilers required.—The number of boilers required at a colliery will depend upon the class of boiler used, the kind of fuel supplied, and the number of engines to be simultaneously supplied with steam.

A good Lancashire boiler, consuming about 7 cwts. of good coal per hour, should suffice for an engine of about 200 horse power, but at collieries where the fuel supplied for firing is often of a very inferior quality, about 180 horse power would be furnished per boiler, or even less. One spare boiler should be allowed for every four or five in use.

An approximate method of estimating the boiler requirements is to allow one 28 ft. × 7 ft. Lancashire boiler for every 13000 cub. ft. of steam required per hour.

Boiler Chimneys.—Chimneys or stacks may be either square, octagonal, or circular, the circular form being the best shape, but at the same time more expensive to build, as the bricks require to be specially made for each section of the chimney. Plenty of area should be allowed in chimneys, as it is useless to make large flues at the boilers and not to have a corresponding area in the chimney. The area at the top should be about the same as at the boiler flues.

The following formula may be employed for chimneys, taking the consumption of fuel at 21 lbs. of coal per sq. ft. of fire grate per hour as a basis:—$A = \frac{1 \cdot 5G}{\sqrt{H}}$ or $\sqrt{H} = \frac{1 \cdot 5G}{A}$; where A = area of chimney in sq. ft. at top or smallest part; G = area of fire grate in sq. ft.; and H = height of chimney in ft. above fire bar level.

If the coal consumed is likely to be lower than 21 lbs. per sq. ft. of fire grate per hour, then the formula $A = \frac{\cdot 07W}{\sqrt{H}}$ may be employed, W being the actual weight of fuel consumed in lbs. per hour.

The height of a chimney will depend on the number of boilers and the total coal consumed, and is often determined by local considerations, especially if the colliery is situated near a town, as the authorities then often specify a certain minimum height for chimneys. Chimneys, as a rule, should not be much less than 90 or 100 ft. in height.

* The following table gives the height of chimney according to the weight of coal consumed:

Weight of Coal consumed per hour.	Height of Chimney.	Weight of Coal consumed per hour.	Height of Chimney.
100 lbs. and under	60 ft.	3000 lbs. and under	160 ft.
500 ,, ,,	100 ,,	4000 ,, ,,	180 ,,
1000 ,, ,,	120 ,,	5000 ,, and upwards	200 ,,
2000 ,, ,,	140 ,,		

The chimney should be built up on good solid foundations, rock, if possible, with a solid bed of concrete 2 to 3 ft. thick for a base, the brickwork being built up vertically for a certain distance and gradually tapered off in 4½ in. courses to the required size. The thickness of brickwork for chimneys depends on their height, but except for very small chimneys it should never be less than 9 in. at the top. The outside batter should be from $\frac{3}{16}$ to ¼ in. per rising foot, or about 1 in 56.

Banking-Out.—Probably few things have undergone more changes in recent years than the banking-out arrangements at collieries. It

* *The Practical Engineer's Pocket Book*, 1897, p. 67.

was, and is still at a large number of collieries, the general practice to cover the landing-stage with flat-sheets or iron plates on to which the tubs could be drawn when taken from the cage, and turned in any direction required. No doubt this system has its advantages, but it requires a great deal of hand labour that might otherwise be dispensed with. When new collieries are being laid out, flat-sheets are now often dispensed with altogether, and lines of rails laid up to the cage.

Fig. 468 shows a banking arrangement on this principle, laid out for dealing with a large daily output.

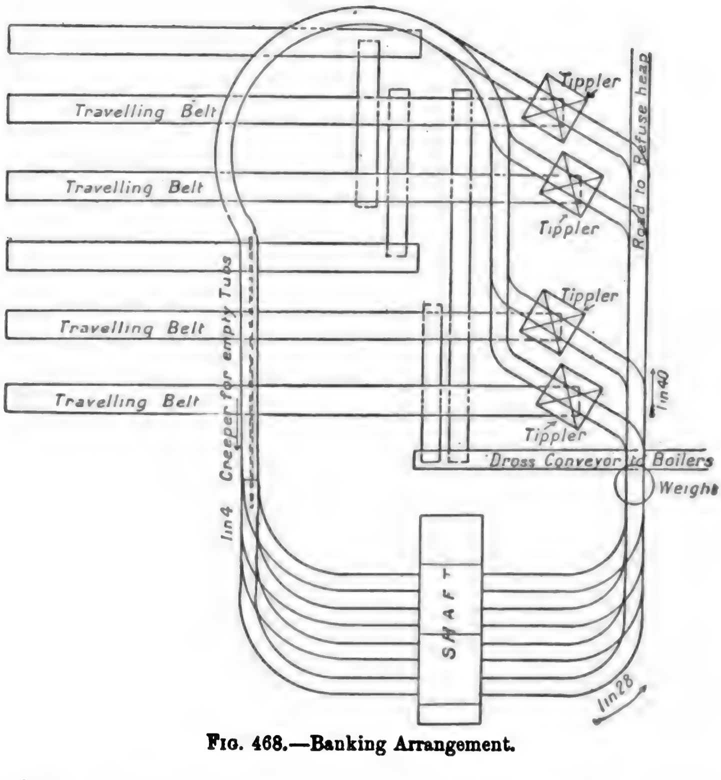


Fig. 468.—Banking Arrangement.

The successful working of such arrangements has been greatly assisted by the introduction of tipplers and the creeping chain. As the lines of rails from the cage on the full side must have an inclination towards the screens, to allow the tubs to run freely, there must be a corresponding inclination against the tubs on the empty lines of rails, which would involve labour to bring the tubs to the cage if

done by hand. To avoid this, the creeper chain is used. The creeper consists of an endless chain made on much the same principle as a travelling belt. At set distances apart are fixed upright pieces or 'fingers' for catching the tubs and carrying them forward to the desired position. The chain works round octagonal wheels at each end, and is supported throughout its length by rollers fixed in standards. On the side of the pit-head where the full tubs are taken off, the rails, as already stated, are laid at an incline in favour of the tub. The tubs are first taken over the weighing-machine, and then by their own weight run into the revolving tipplers, from which, when empty, they pass to the creeper and are hauled up to the level of the cages again. Instead of a 'creeper' a steam or hydraulic hoist is sometimes used to raise the empty tubs to the desired level. The disadvantage of such an apparatus is that it is not self-acting, and is therefore more costly in working. The hoists themselves are also expensive to erect, especially if the height is great, and the raising piston must of necessity be long in proportion.

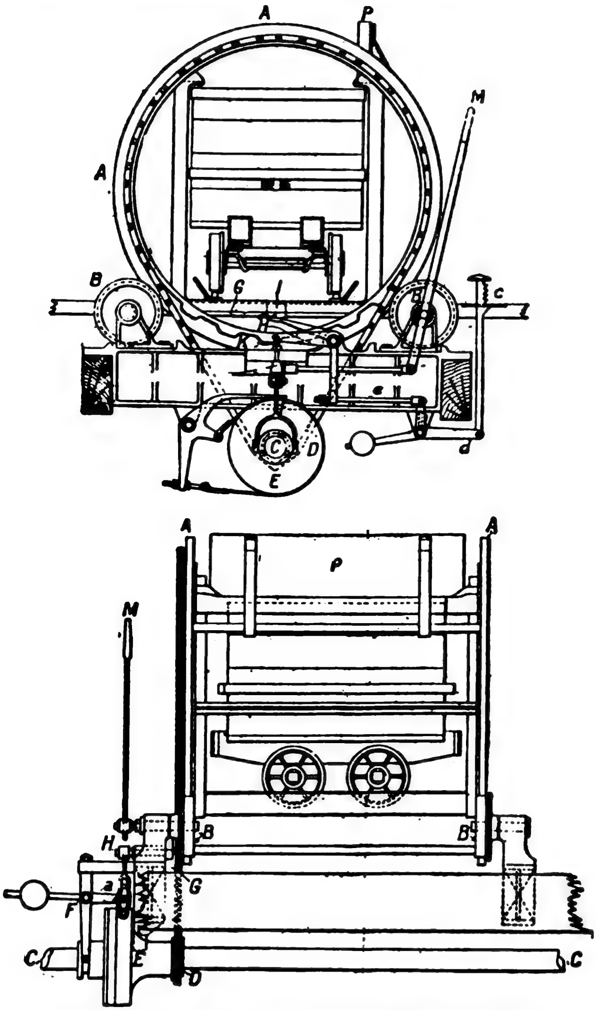
Tipplers.—The simplest and most common kind of tippler for emptying the contents of the tubs on to the screens used to be an arrangement of two rails bent up in the end, and pivoted on an axle placed beyond the centre of gravity from one end, so that, when the tub was run to it, it was tipped up and emptied, and had then to be drawn back into its original position. This tippler is now seldom adopted, as the coal had to fall too great a distance down to the screen, and also necessitated the tubs being provided with a door at one end for emptying, which is a disadvantage for underground work, as such doors are apt to fly open and allow the coal to fall on the road.

Rotary Tipplers.—During recent years a large number of tipplers have been designed, chiefly on the rotary principle, in which the tub is turned through either a half or full circle.

One of this class is Wood & Burnett's, which is shown in figs. 469, 470. It is driven by a chain and sprocket wheel arrangement, or by toothed gearing, the starting and stopping gear being the same in either case.* The tippler consists of circular cast-iron wheels A, riveted together by bars and angles in the usual way, and supported on four cast-iron rollers B. A counter shaft C, driven by the screen engine, carries the sprocket wheel D, attached to a friction clutch E, by means of which the tippler may be thrown into gear with the shaft C. The friction clutch also acts as a brake-wheel, a wrought-iron strap passing round it for that purpose. The sleeve of the friction clutch and the brake-lever are moved by means of a weighted bell-crank F, so arranged that when the end *a* is depressed the brake is applied and the friction clutch thrown out of gear. On releasing it the weight causes the brake to be released, and the friction clutch gears on again. The tippler is started by means of

* *Trans. I. M. E.*, vol. ix. p. 232.

the hand-lever M, so that the wedge L is withdrawn from the bolt K; the latter is thus shortened, allowing the weight on the lever F



Figs. 469, 470.—Wood and Burnett's Rotary Tippler.

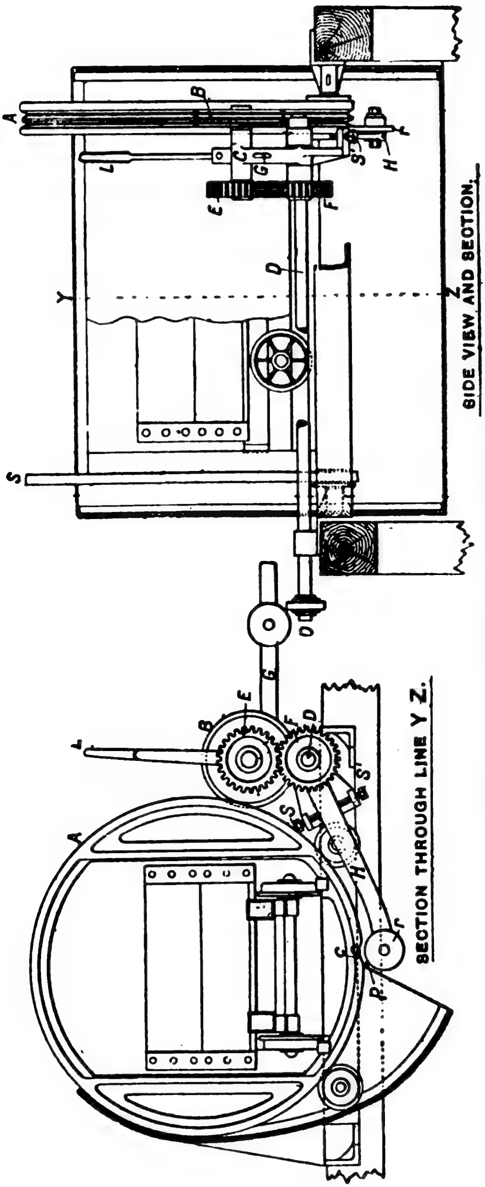
to fall, by which the brake is released and the friction clutch put into gear. At the same time the lever O falls clear of the stop I, and the tippler at once commences to rotate.

The hand-lever M is then allowed to fall back and to push the wedge L through the aperture in the box, thus lifting the end of the lever G into position ready to engage with the stop I, when the revolution of the tippler has been nearly completed. The lever G is pushed down by the stop I, and so applies the brake and throws the friction clutch out of gear automatically and simultaneously.

An auxiliary attachment, consisting of a treadle, lever, and rods *c d* and *e*, is applied, so that the tippler can be stopped during any portion of its revolution, if necessary.

Heenan and Froude's Rotary Tippler.—This tippler is also of the rotary type, having the cylindrical frame supported upon two pairs of rollers. The cast-iron ends A *s* (figs. 471, 472) of the tippler are dissimilar, A having two grooves on the edge, so as to engage the friction wheel B when in motion. This friction wheel is mounted on an arm C, which is pivoted on to the countershaft D, from which the motion is transmitted to the friction wheel by the spur wheels E and *F*. The countershaft D is driven from the screen engine, and is in constant motion. The arm C also carries a hand-lever L, by which the friction wheel is placed in contact with the grooved rim A of the tippler frame. An arm G, with a counterweight attached, is used to throw the friction wheel B out of gear, and to automatically stop the tippler at each end of the revolution. The lever H, having a small roller *r* and a pawl *p* mounted on one end, is pivoted loose on the shaft D, and its movement is transmitted by means of the adjusting screws *s* and *s′* to the arm C, and thence to the hand-lever L and friction wheels. After placing a tub on the tippler, the banksman pushes over the lever L, so that the roller *r* and the pawl *p* are lifted from the recess *c* on the rim A, and the friction wheel B is put into gear at one movement. On completing a revolution, the roller *r* drops into the recess *c*, throwing the friction wheel B out of contact with the rim A, and the pawl *p* brings the tippler to rest in the proper position for receiving another tub.

Travelling Belts.—After the coal has been passed over the screens to get rid of the small coal or dross, it is received on a travelling belt, where it may undergo a final hand-picking before being loaded into waggons. The belts are made of varying materials, according to the work to be done. For coal picking and conveying they are usually of iron or steel plates, link or woven wire, or of flat hemp or cotton belting with a hardened indiarubber face. Where only two kinds of coal have to be separated from each other, such as a cannel and a free coal, belting made of canvas faced with leather can be used with much advantage. The ordinary type of belt is made of steel plates 9 to 12 in. broad, $\frac{1}{8}$ in. to $\frac{3}{16}$ in. thick, fixed to iron or steel link chains that fit into a split polygonal drum at each end. The belt is supported usually on cast-iron rollers keyed to a shaft, which runs in pedestals fixed to the supporting beams. The delivery end is provided



FIGS. 471, 472.—Heenan and Froude's Tippler.

with tightening screws and slotted seats to take up any slack on the chain (see fig. 473).

In driving, the usual type of hexagonal wheel is employed, two being placed at each end of the belt. These belts have usually a very slow motion, travelling at the rate of 40 to 80 ft. per minute. The length of the belt may be anything between 30 and 150 ft., but where they are used for picking the dirt out of the coal 30 to 40 ft.

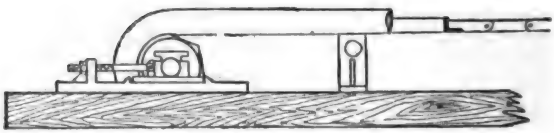


FIG. 473.—Tightening Screws for Belt.

is a usual length. Their width varies from 2 to 4½ ft., 3 ft. or 3½ ft. being a good width where the picking is done from one side only. These belts are now often of steel wire rods woven across each other, as shown in fig. 474, but this type of belt can only be used for large sizes of coal.

Elevators.—When the small coal or dross is dropped through the screens it generally falls into a large hopper or dross pit, and is thence raised by means of elevators to be further treated either by dry or wet cleaning. These elevators are made on the same principle as the travelling belt. Two long endless chains with long single links fixed alternately inside and outside each other, having an iron bucket fixed to them, pass round a hexagonal wheel at each end. The length of these elevators and their speed depend greatly upon the quantity of coal to be handled, and the height to which it requires to be raised. An average speed is 300 to 400 ft. per minute.

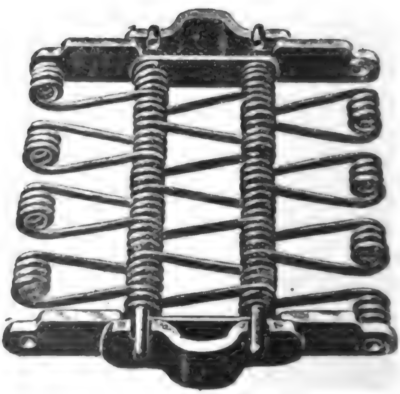


FIG. 474.—Segment of Wire Belt.

Arrangement of Cleaning Apparatus.—We have described the various parts of a coal-cleaning arrangement, either of which is seldom used by itself, but all of them being used in combination, whether the coal is subjected to a dry or wet cleaning process or not.

If the coal can be cleaned sufficiently well without washing, the arrangements are greatly simplified, and less outlay in plant is

required. Fig. 475 illustrates an arrangement of an ordinary cleaning plant. It must be remembered, however, that a given arrangement of plant will hardly ever suit two different collieries, as the screening arrangement will altogether depend on the quality of the coal, the amount of dirt in the coal, and the kind of coal in demand.

Coal Cleaning.—Under this heading may be included the various methods of washing and sorting the coal when brought to the surface. The processes employed may be divided into two groups:—

Cleaning by mechanical means;
Cleaning by taking advantage of the physical properties of the mineral.

Under the first head may be included (*a*) screening; (*b*) washing the mineral to cleanse it from mud, clay, shale, etc.; (*c*) hand-picking; and (*d*) sorting.

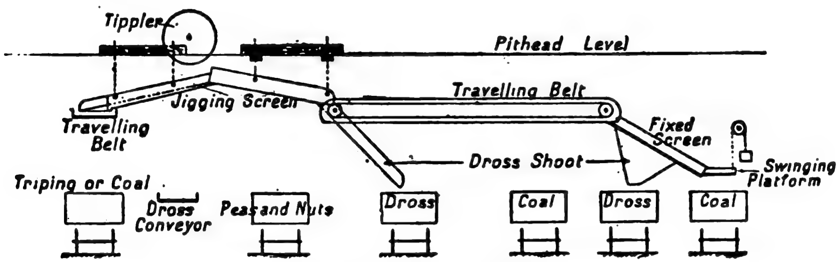


FIG. 475.—Cleaning Plant.

Screening the Coal.—This is usually the first operation to which the coal is subjected after it arrives at the surface. The screens employed may be divided into classes, viz.,

Fixed inclined screens, with movable bars.
Revolving screens.
Jigging or shaking screens.

Fixed Inclined Screens are very largely used, but they are fast being superseded by the shaking type of screen. When the former only are in use, the coal is picked by hand, and, as a rule, this is the only cleaning it undergoes. Fixed inclined screens are usually made of iron or steel bars set at an angle of about 30° from the horizontal, so that the coal can just move slowly forward. The bars may be made of different shapes, but are very often rectangular in section, about 1 in. thick, 3 to 4 in. deep, and 10 to 15 ft. long, according to the quality of coal dealt with. A soft coal generally requires a longer screen than a harder quality, as there is more dross to be got rid off. The openings between the bars vary from $\frac{1}{2}$ in. to $1\frac{1}{2}$ in.,

according to requirements, and sometimes by agreement with the landlord of the coalfield, as different 'lordships' or royalties are often payable on different qualities or sizes of coal.

Revolving Screens.—These screens are made like a large revolving riddle fixed on a central axis. They are largely used for dry-cleaning and sorting the smaller qualities of coal after they have been passed over a fixed or jigging screen. They may be made either of wire netting or sheet-iron plates with circular perforations to suit the size of coal required. A revolving screen is usually made to sort or separate two or three different sizes of coal, different parts of the screen being provided with differently sized apertures, as shown in fig. 476.

Jigging or Shaking Screens.—This class of screen is now almost exclusively largely used in place of the fixed bar type for the screening and sizing of coal. The screens are set at very slight inclination, 10° to 12° from the horizontal, and hence the coal descends very

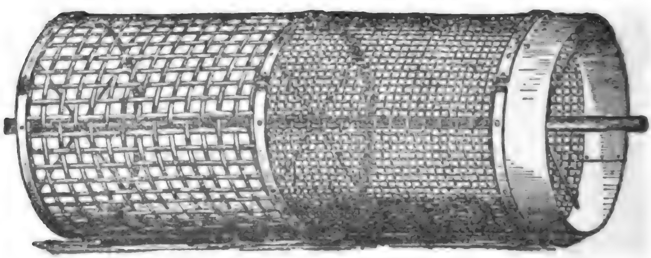
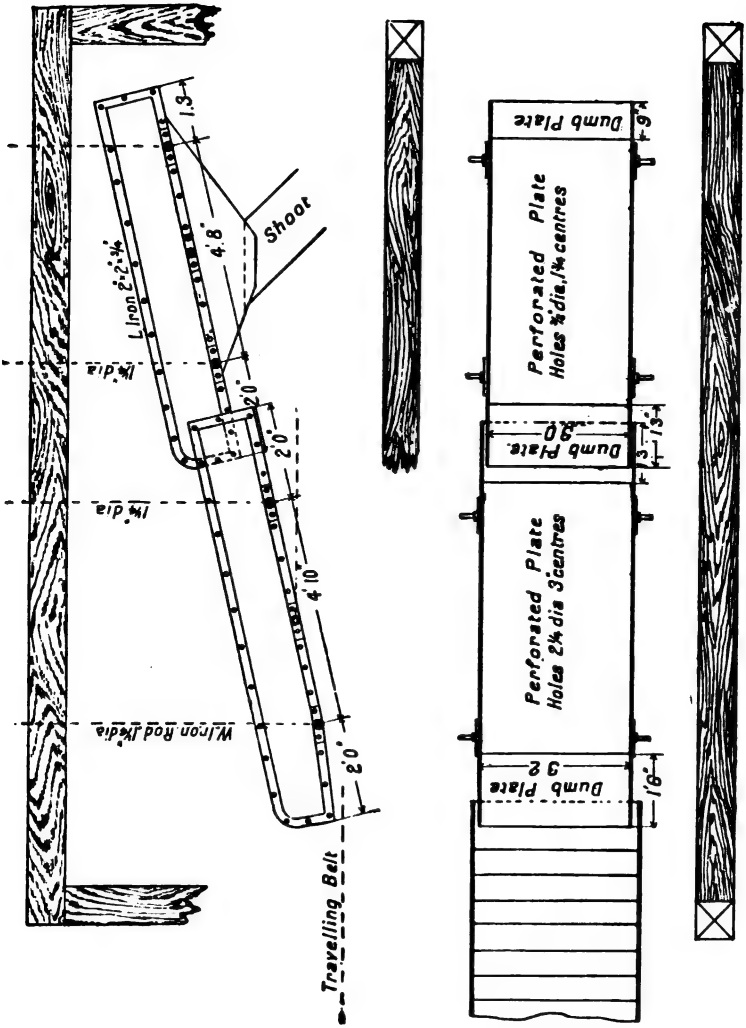


FIG. 476.—Revolving Screen.

slowly, gets very little broken during the process, and can be thoroughly picked and cleaned. The screens usually derive their motion from arms attached to the sides, and which work eccentrically, giving the screens a rapid to-and-fro motion, the coal at the same time moving slowly forward. They are usually suspended to beams by means of four rods fitted with adjustable screws for altering their inclination if required. The eccentric for driving has a short stroke, 3 to 6 in., and makes about 60 to 100 revolutions per minute. The screens may be constructed of wire netting fixed in a frame, or of sheets of iron or steel with circular openings. The latter are very largely employed, and are preferable to those constructed of wire netting, as they last much longer and size the coal much better. Figs. 477, 478 illustrate the construction of a jigging screen and the method of fixing it.

Coal Washing or Wet Cleaning.—Coal below a certain size, when mixed with foreign substance, is difficult to clean properly without



FIGS. 477, 478.—Jigging Screens.

washing. Some coals are so intermixed with impurities that no other method than wet cleaning would be of any use. As small coal usually contains the highest percentage of impurities, it is usually dealt with in this way. The cleaning of coal may be performed in many ways, all of which are, however, based on the same principle, *i.e.* the differences in density existing between the impurities and the coal. The different methods of washing coal are*: (1) by an ordinary inclined trough washer with fixed stops; (2) by a fixed trough washer with movable scrapers, of which Elliot's patent is an example; (3) by movable troughs, such as the Murton and the Wood and Burnett washers; (4) by continuous ascending current washers, such as the Robinson washer; (5) by intermittent ascending current washers, as in the Lührig and Coppée washers; (6) by compressed air, as in the Baum washer.

Choice of Washer.—In choosing a washing plant the most important consideration is the financial one, and this is generally where the difficulty lies, for a certain class of machine may be expensive in first cost and yet may soon recoup the outlay by economy in working and in producing a cleaner and better quality of coal, while a machine that may be erected at half the cost may prove much more expensive in other respects in the end. An important point also is that the coal must be adapted to meet the buyers' requirements, while different kinds of impurities require different methods of treatment. M. Callon states the position clearly when he says:—"In the case of coal cleaning we are dealing with enormous quantities of a substance of comparatively little value, the profit on which is reduced to a minimum by competition. We are consequently forced to employ special machines as simple as possible in construction, and capable of treating considerable quantities. We must abandon all idea of treating coal by frequent repetitions of the same process, or, at all events, do so sparingly, so as not to burden the undertaking with heavy costs, and the operations must be carried on with simplicity and economy."

For small quantities of coal, and where not much sorting is required, the trough washer is often found satisfactory, or, if a more thorough cleaning is desired, the ordinary 'Bash' washer is as good as any for economical working.

When large quantities have to be dealt with, and the coal requires sorting into a variety of sizes, some of the more expensive machines, such as the Lührig or Coppée, may be adopted. Before any class of machine is fixed upon, the coal and its accompanying impurities should be submitted to a thorough analysis, the result of such a test often affording sufficient information for a decision to be arrived at. The three important points in coal cleaning are:—To remove impurities as far as possible; not to allow any coal to pass away

* *The Practical Engineer's Pocket Book*, 1897, p. 323.

with the impurities; and to achieve these objects in the cheapest manner with efficiency.

Trough Washer.—This is one of the earliest and simplest types of coal washers. It consists of a long, narrow trough divided into a number of stages by means of projecting pieces fixed to the bottom or sides of the trough. The trough is set at an inclination of 2 in. or 3 in. per yard, and a stream of water is allowed to flow continuously and sufficiently fast to carry the coal forward over the projections, while the heavier impurities fall to the bottom and are caught in the divisions of the trough. When a certain quantity of coal has been cleaned the flow of water is stopped and the refuse cleaned out, ready for another operation.

An improvement on this form of trough, with stationary dams, is a travelling scraper on the endless-chain principle, which by moving along the bottom of the trough against the stream of water and coal, delivers the impurities automatically at the upper end of the trough. The dimensions of these troughs are from 50 to 80 ft. long, 2 to 3 ft. wide, and 10 to 12 in. deep. The trough washer is best suited for small quantities of coal; it requires a large flow of water and entails extra labour, but it has the recommendation of being simple in construction and cheap at first cost. A trough washer 60 ft. long and 2 ft. broad, with revolving riddle and connections, can be erected for about £180.

Elliot Trough Washer.—This is an improvement on the old trough washer. It consists of a wrought-iron or steel trough, about 18 in. wide at the bottom and 30 in. wide at the top, and having sloping sides. At each end a sprocket wheel is fixed, round which an endless chain passes, and attached to the chain at intervals of 6 ft. are fixed scrapers to fit the inside of the trough. The scrapers form stops or dams, which are slowly moved by the chain along the trough in the opposite direction to the flow of the water. The trough is set at an inclination of about 1 in 12, and the coal is admitted at the centre of its length and the water at its upper end. As the water runs down it carries with it the coal, which is lighter than the dirt, while the latter settles in the scrapers and is carried against the stream of water and delivered at the opposite end.

Murton Coal Washer.—This machine, like the Elliot washer, is an improved trough washer. It consists (fig. 479) of an endless articulated steel trough belt, which is watertight. This trough revolves slowly round suitable drums at each end, the action being continuous and automatic. The clean-washed coal falls into a hopper at the lower end, the dirt or refuse is delivered into a dirt hopper at the upper end. The trough is constructed of steel, 60 ft. long, 3 ft. wide, 8 in. deep, and fixed at an inclination of 1 in 18. The supporting drums and rollers are mounted on a suitable frame. Inside the trough, at intervals of 3 ft., are dams or stops about 2 in. high.

Method of Working.—The trough is set in motion and travels up

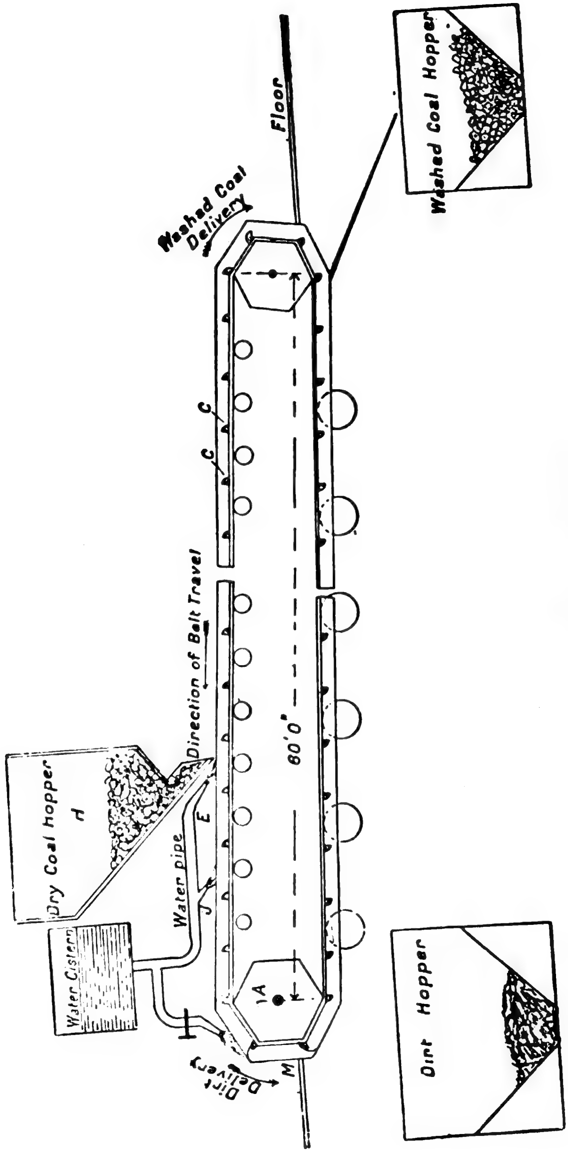


FIG. 479.—Murton Coal Washer.

the incline, and towards the coal and water supply, at a speed of from 8 to 10 ft. per minute. The coal from the hopper H (fig. 479) and water from the nozzle E are turned on in suitable proportions. The latter, carrying the coal with it, flows in a direction contrary to that of the trough to the coal delivery end D, where it is delivered free from impurities. The dirt falls to the bottom of the trough, and is arrested by the stops *c c*, and carried forward by the motion of the trough beyond the point E, after passing which it is agitated in such a manner that any coal mixed with it is liberated, and carried back by a strong current of water from the supply pipe J to F, where it joins the regular coal feed, and is carried along with the rest of the clean coal forward to D. The refuse continues to be carried upwards to the end of the trough M, and as the belt moves over the drum A it falls off into the hopper. The trough is washed clean by the water spray *s*, and passes to the drum A′ at the lower end D. The coal, when delivered at the lower end, falls into a spout, with a draining plate (with 125 perforations $\frac{1}{16}$ in. diameter per sq. in.), and the water is quickly drained away. About 450 gallons of water per minute are required when washing 400 tons of coal per day, but the same water can be used over and over again.

Robinson Washer.—This machine, which is shown in fig. 480, consists of a truncated inverted cone, made of steel plates about 8 ft. diameter at top and 2 ft. diameter at bottom and $6\frac{1}{2}$ ft. deep. A strong shaft is fixed vertically, to run in the centre of the cone; on this shaft is a strong cast-iron cross-head, to which are bolted four cross arms. To each of these cross arms are bolted three heavy wrought-iron bars, but outwards at the bottom, and projecting down so as almost to touch the sides of the cone or washer, and on the bottom of the driving shaft, are also bolted four shorter arms, as shown in illustration. In practice the coal passes into the washer from the spout A into the centre ring B, while the water supply is forced in at the bottom at E and through the perforations G.

The coal is kept in a continual state of agitation, and as it sinks into the washer it is met by the upward current of the water; and, being lighter in weight than the impurities with which it is associated, is floated upwards, as indicated by the arrows, and overflows at D. The impurities sink downwards and are collected in the chamber J. When this chamber is filled, the upper valve H is closed, and the lower valve H′ opened; this allows the accumulated refuse to be discharged, after which the lower valve is closed and the operation repeated. The shaft, with cross-head and depending arms, revolves at the rate of 14 or 15 revolutions per minute.

In fig. 481 is shown a general arrangement of a Robinson washing plant.

The clean coal passes from the washer on to inclined perforated screens or shoots, and is here freed from the water, after which it is either carried by means of conveying machinery to storage bins, or

falls direct into the waggons. The water from the overflow is collected in settling ponds, and by means of a pulsometer pump is again forced into the washer for further use.

The Robinson washer is cheap to construct and maintain, and requires little water, but it largely depends for its efficiency on the attention and skill of the man in charge, who may often be tempted to pass more coal through it than it can effectually deal with. These

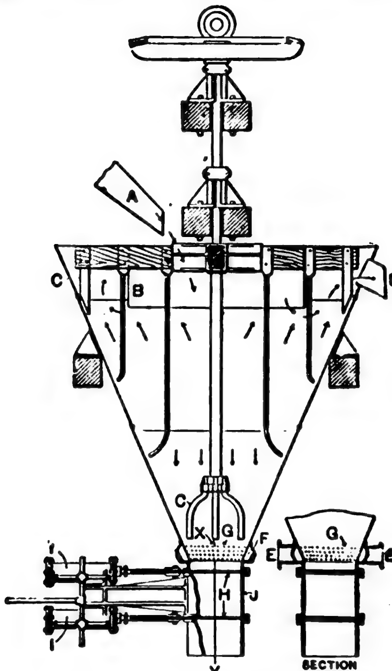


FIG. 480.—Robinson Washer.

machines can be made to wash from 20 to 40 tons per hour. The cost of a Robinson washing plant to treat 20 tons per hour is from £250 to £350.

Lührig and Coppée Machines.—A paper read before the South Wales Institute of Mining Engineers * gives an elaborate description of the construction and working of these machines. The following abridgment † will, however, suffice to explain their action. The machines are made in two sizes, the larger dealing with coal from 5 in. to $\frac{3}{8}$ in. in thickness, and the smaller taking fine coal and powder.

* *S. Wales Inst. Min. E.*, xiv. pp. 88-102.
† *Hughes's Coal Mining*, pp. 416-4.

"The larger, or nuts machine (figs. 482, 483), is of the ordinary continuous jig type, and consists of two compartments A and B, in one of which the piston works, while the other is provided with a perforated strainer, slightly inclined from front to back. The piston P receives an up-and-down motion by being connected to cranks on a horizontal shaft, and the amount of this throw can be varied from $1\frac{3}{4}$ to 4 in. An opening W runs along the front of the washing compartment, and through this clean coal continuously passes away. The shale is discharged through a small cylindrical compartment D, connected to the

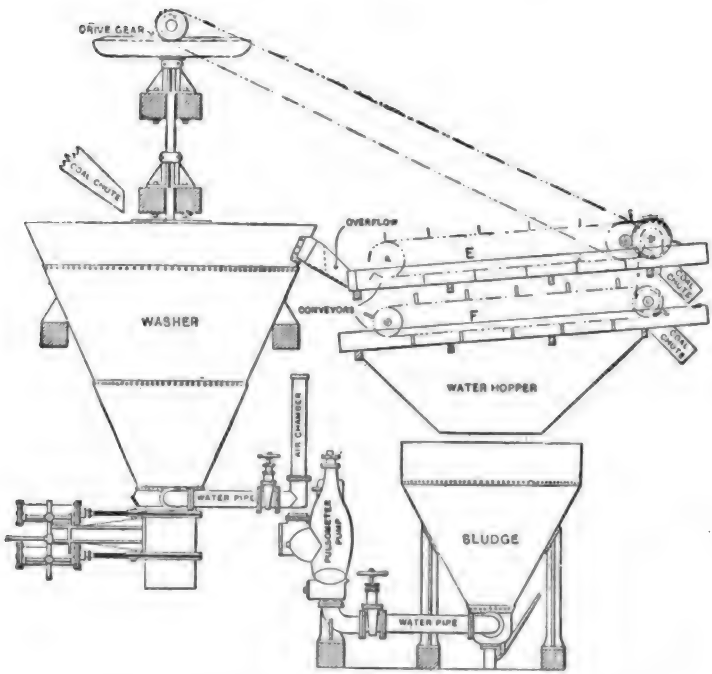
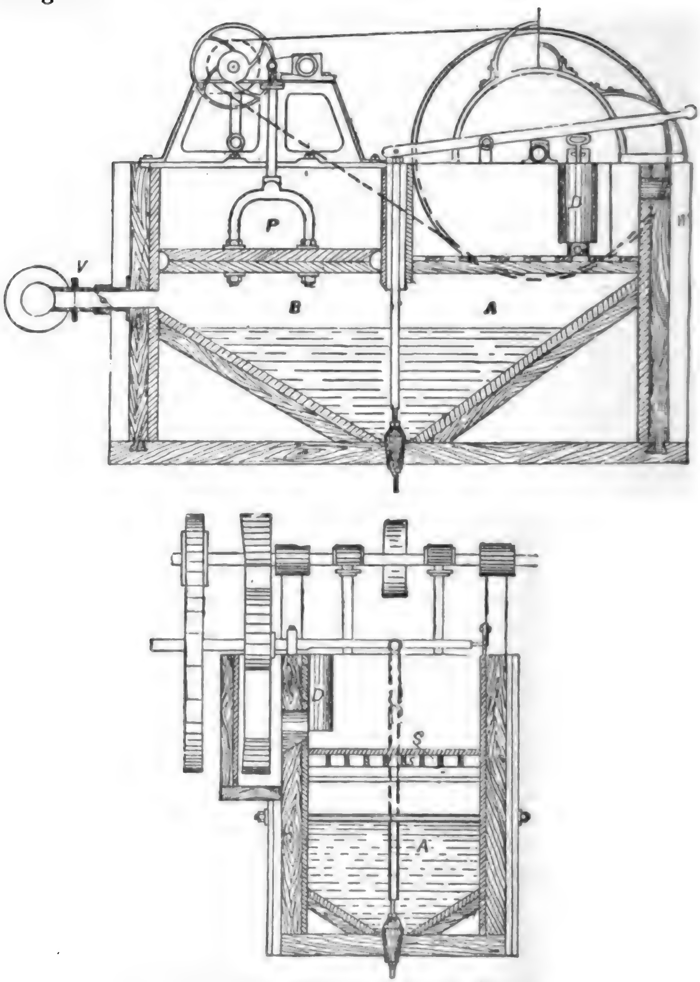


FIG. 481.—General arrangement of a Robinson Washing Plant.

side of the casing, but which starts above the level of the strainer, leaving a free space between the strainer and the lowest end of the compartment of about 3 in. It is open at both ends, and communicates with the outside of the machine through the opening R. It is provided with a sliding door which regulates the discharge of the shale.

"When the unwashed coal is introduced into the machine, and the piston descends, it drives water into the compartment B, and lifts the bed of the material resting on the strainer. On the return stroke, the heavier dirt falls faster than the lighter coal, while in the upstroke

the lighter coal is lifted farther than the heavier dirt; the result is, that the two substances separate into layers, the coal being, of course, the higher.

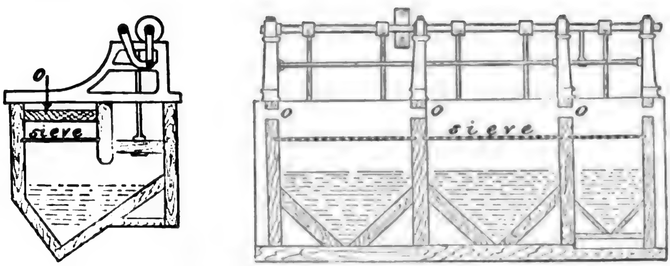


Figs. 482, 483.—Lührig and Coppée Machines.

"*The Felspar Washer* is of similar construction, but differs materially in its method of working. It consists of a box, divided into two compartments by a longitudinal partition, in one of which the piston works as before (figs. 484, 485). It is also generally divided into

two or sometimes three compartments in the direction of its length, each communicating with the other by openings *o* along the side, and through these the washed coal passes away. In the nuts washer the holes through the sieve are smaller than the size of the material being treated, and consequently no discharge takes place through them. In the felspar machine they are larger than the material, and the dirt passes through the sieve into the lower part of the apparatus Three sieves are generally employed. The dirty coal is introduced at one end and gradually passes down over the remaining gratings, the clean material being finally discharged at the opposite end.

"The chief peculiarity is the introduction of a layer of felspar, from 2 or 3 in. thick, on each sieve, whose specific gravity is greater than that of the material to be concentrated, and yet less than that of the gangue. The sizes of the particles of this bed are larger than the holes in the sieve. The whole framework of the machine is



Figs. 484, 485.—Felspar Machines.

filled with water up to the level of each sieve, and as the pistons work up and down, a volume of water is forced through the holes in the bottom of each sieve, lifting the bed and the layer of material on it, and then allowing the whole to fall again on the return stroke. The lighter coal rises to the surface, and the heavier dirt gradually finds its way through the bed of felspar, when it falls into the bottom of the compartment to be removed from time to time. It is essential for thorough cleaning that the size of the felspar should be as small as allowable, and that the particles of mineral forming the bed should be of convenient density, have well-defined rectilinear angles, and be of great durability to resist wear and tear. A point of considerable importance is the proper regulation of the delivery of water, which is controlled by a tap; upon this depends the progress of the material and the time it is operated upon.

"For very dirty coal, perhaps no machine does its work so efficiently as this; indeed, everyone gives it the character of removing dirt. It is, however, expensive in the first cost, but requires little attention. Much depends upon the percentage of dirt originally present in the coal. If it is small, and, say, one-half of it is removed, the coke from

the resulting product is a fair one; on the other hand, where the dirt amounts to from 15 to 30 per cent. and only 3 to 10 per cent. is taken away, the coke is very bad. With a dirty coal, probably it is best to use machines of this type."

Baum Washer.—This machine applies a new principle to coal

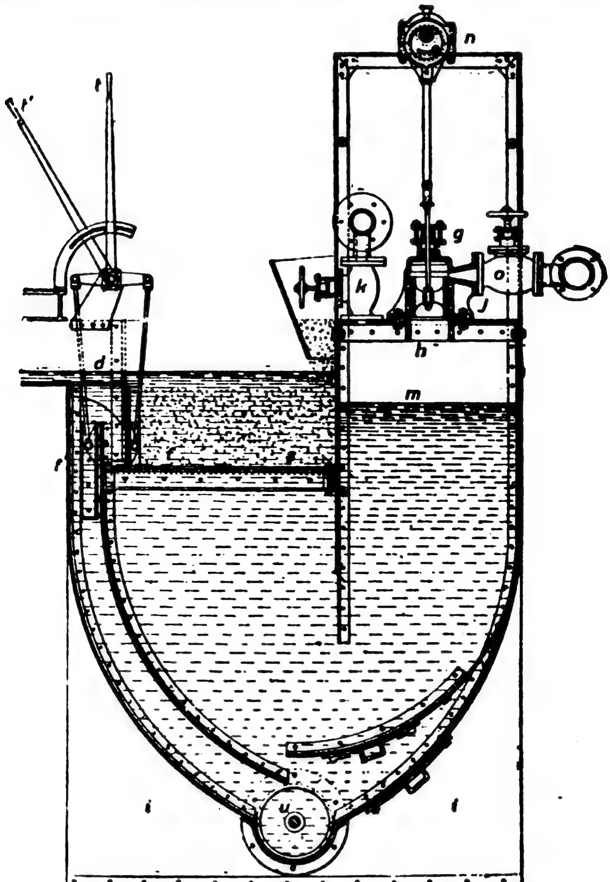


FIG. 486.—Baum Washer.

washing, inasmuch as compressed air is used, instead of a vertically reciprocating piston, for producing the oscillations of the water through the bed of the jigger. By this method it is claimed that all noise, vibration, and shock are done away with, and that less power is required than in an ordinary piston machine. Fig. 486 shows a

cross section of one of the jigging machines.* The jiggers are constructed of steel or wrought-iron plates connected by angle irons. The water-box *m* is closed at the top by a horizontal plate, to which is fitted a piston valve casing *g*, in which works a piston or cylindrical valve *h*, actuated by the eccentric *n*, and having ports or openings in the sides to control the admission and exhaustion of compressed air to and from the upper closed-in part of the water-box *m*. Openings *j* are provided in the sides of the casing *g* for the escape of the compressed air from the water-box; and *o* is a stop valve for admitting or shutting off the compressed air. The latter, at a pressure of 1½ to 2 lbs. per sq. in., is drawn from the receiver, to which it is supplied by the air compressor.

In the coarse jigging machines the valve *h* makes from 50 to 70 strokes per minute, and in the fine jiggers from 75 to 110 strokes per minute, the alteration in speed being effected by means of coned pulleys on the driving and countershafts. The jigger sieve *s* is of ⅜ in. mesh for the coarse and ¼ in. mesh for the fine jiggers. At the top of the machine there is an overflow bar *d*, over which the coal from the coarse jiggers is carried along channels to special nut pockets; *f f'* are two slides, adjustable by means of the levers *t t'*, for regulating the outflow of the dirt, which falls to the bottom of the jigging-box, and is conveyed away by means of a spiral conveyer *u* extending the whole length of the jig battery.

It will be noticed that the mesh used in the fine jigging machines is larger than much of the material treated, and it might be supposed that some of the latter, including the coal, would fall through the sieve. The machines, however, can be so regulated that the larger pieces of stone or shale are retained a considerable time on the sieve, and owing to the weight of material above them they set themselves parallel to the sieve and form a bed similar to the felspar bed employed in the Coppée fine jigging machines.

Cost of Coal Washing.—This will vary very much with the class of machine used, the quality of the coal, and the percentage of dirt accompanying it; and also on the amount of dressing or preparation the coal requires for the market.

The following table † shows the cost per ton of washing by various machines in the year 1886:—

Name of Colliery.	Dry Cleaning.	Wet Cleaning.	Type of Washer.	Condition of Coal previous to Washing.
Barrow, . .	1½d.	2½d.	Robinson.	Crushed.
Aldwark Main, .	...	2⅝d.	Trough.	,,
Nunnery, . .	3d.	1·40d.	,,	Not crushed.
Annesly, . .	2½d. to 3d.	...	...	...
Clifton, . . .	3d. to 4d.	6d. to 7d.	Coppée.	Not crushed.
North Motherwell, .	...	0·7d.	Lührig.	,,

* *Trans. I.M.E.*, vol. vii. p. 158.

† *Trans. M. I. Scot.*, Coal Cleaning Committee's Report, p. 179.

CHAPTER XVI.

SURVEYING, LEVELLING, AND PLANS.

"SURVEYING is the art of ascertaining by measurement the shape and size of any portion of the earth's surface, and representing the same on a reduced scale, in a conventional manner, so as to bring the whole under the eye at once." *Surveying has also been defined by Mr Bennett H. Brough as "the art of making such measurements as are necessary to determine the relative positions of any points on the earth's surface. From such measurements a plan of any portion of the earth's surface may be drawn, and its area calculated."

In ordinary mine surveying three objects are in view, viz., to obtain a correct plan and section of the underground workings; to ascertain the correct position of all surface buildings, shafts, and other features, and the correct boundary line of the field or royalty to be worked, and to connect as accurately as possible the workings on the underground plan with the surface survey.

† "All surveys are conducted on nearly the same principles, the difference consisting in the style of the instruments used in the work, and the different methods of calculating the various data connected with the survey."

The instrument which is in most common use for ordinary underground surveys of mines is the miner's compass or dial, sometimes called the circumferentor. This is an instrument the chief use of which is to measure horizontal angles. As used with the loose needle the angles are measured in relation to the magnetic meridian, and as used with the vernier or fast needle the angles are measured either in relation to the magnetic meridian or in relation to any given base line. The instrument consists essentially of two parts, a needle swinging freely on a pivot, and the dial on which the angles are read. When surveying with the magnetic compass it is most important to remember that the needle does not point to the *true north*, and that the direction in which it points is subject to several variations from

* *A Treatise on Mine Surveying*, by Bennett H. Brough, p. 1.
† *Ibid.*

time to time. It is more than 300 years ago since observations were first made on the variations of the needle, and we are therefore not without some data to guide us. In Queen Elizabeth's time, *i.e.* about 1580, the variation was about 11° 11′ east; in Charles II.'s time, 1657, this variation had disappeared altogether and the needle coincided with true north. The needle swings backwards and forwards on each side of the true north like the pendulum of a clock, but with this difference, that it swings in centuries and not in seconds. Its maximum variation is 24° 48′ on either side, and this is termed the *secular variation.* Taking the average between 1580 and 1880, the variation was 8¼ minutes per year, and for the ten years between 1877 and 1887 the total variation was 1° 12′, giving an average of 7·2′ per year. Since 1660, in which year there was no variation, the declination has been towards the west, attaining its maximum in 1820, and then gradually decreasing, so that at the present time the magnetic meridian is again approaching the true meridian. The average annual variation of the needle may be taken at 8 minutes.

The magnetic needle at Greenwich at this date (1899) points about 16° 30′ to the west, as shown in fig. 487. At Edinburgh the magnetic declination is 3° greater than at Greenwich, while in Glasgow and Dublin it is 3° 50′ greater. In the North of England the variation is about 1¾ degrees greater than at Greenwich.

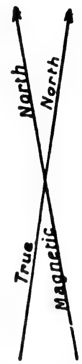


Fig. 487.

A very elaborate magnetic survey was carried out between the years 1884 and 1891, by Professors Rücker and Thorpe, to determine the amount of variation and the lines of attraction in the United Kingdom. Observations were made at 882 different places, and from their calculations and a map it will be possible to gain some idea how far local disturbances are likely to affect the needle at any given place.

As regards such disturbances it is found that the north-seeking pole of the compass is attracted to certain regions and to lines which can be traced for scores or even hundreds of miles. These lines they (Rücker and Thorpe) call *magnetic ridge lines.**

The principal ridge lines determined were as follows:

(1) In the Scotch coalfield a ridge line runs from the neighbourhood of North Berwick to the Clyde between Glasgow and Hamilton, turns south to Newmilns in Ayrshire, and finally runs in a northerly direction towards Ardrossan and Arran.

(2) The Yorkshire coalfield is dominated by a ridge line which runs south-west from Harrogate towards Keighley, and follows the outcrop of the Millstone Grit to Matlock.

* *Trans. Inst. Min. Engs.*, vol. ix. pp. 418–419.

(3) In the southern part of the Derbyshire coalfield the north-seeking pole of the magnet is attracted from all sides to a centre which is a little to the north-east of Nottingham.

(4) In the Lancashire coalfield a ridge line runs from Rochdale to Wigan and Southport.

(5) In the South Wales coalfield a ridge line runs from Risca to the south end of Ebbw Vale, and another passes from Brecon to Neath, and thence nearly due west through the centre of the coal measures.

If these ridge lines are therefore drawn upon a map, and the variation determined for places in their vicinity, the magnetic variation for any given locality may be pretty accurately ascertained.

The amount of variation may also be approximately determined for any locality in Great Britain by drawing lines on a map in the following directions: *

(*a*) From Winchelsea (Sussex) by point of Sheppey Island and Ipswich, the magnetic declination is 30 minutes less than at Greenwich.

(*b*) From Cowes (Isle of Wight) by Basingstoke and Great Grimsby 30 minutes more than at Greenwich.

(*c*) From Start Point by Teignmouth, Newport (Mon.), the Peak to Seaham 1°·30′ more.

(*d*) From the Land's End by Great Ormes Head, Skiddaw, etc., 2° 30′ more.

(*e*) From Peel (Isle of Man) by Wigton to Falkirk, 3°·20′ more. If these differences are applied to the declination at Greenwich, the amount of declination or variation on each line may be obtained. For any place situated on one of these lines the value of the variation will thus be at once ascertained, and for a place situated between any two lines a proportional variation may be made from the values found for the adjacent lines.

The variation of the magnetic meridian shows the necessity of recording on all colliery plans the date and amount of declination for the guidance of future working. Many accidents can be traced to the neglect of doing this at the time of drawing up the plan. The date of an old plan may be approximately ascertained by looking at the meridian line on it; thus, if a plan is found having a meridian with a declination of 21° 00′ west of the true north, we would say that it was probably made about the year 1864, the variation for that year having been 21° 03′ at Greenwich. The needle is also subject to diurnal fluctuations.

Diurnal Variation.—In the afternoon the needle is drawn a few minutes to the west and in the early morning a few minutes to the

* *Colliery Manager's Pocket Book*, 1898, p. 151.

east. The needle stands at its mean position a little after 10 a.m. and a little before 7 p.m. The variation is greater in summer-time than in winter, but seldom exceeds about 10 minutes. This variation must be taken into account when making surveys in which accuracy is required, but under ordinary circumstances, such as the quarterly extension of colliery plans, the variation is too small to cause any appreciable error.

The declination of the needle may be due to various causes, such as :—

(1) The presence of masses of magnetic rock.
(2) Induction currents.
(3) Magnetic storms.

The chief mineral that attracts the needle is magnetite, which is found in masses, veins, and sands. Tops of mountains in the Northern Hemisphere attract the north-seeking end of the needle. Beaches are often covered with magnetic sand; for instance, in Rothesay Bay there is a large bed of this description. Other minerals that attract the needle are magnetic pyrites and native iron. All crystalline rocks contain minerals which deflect the magnet more or less, and rocks which are dark-coloured do so more than others, as such rocks usually contain a fair proportion of iron compounds. Sometimes the needle is affected in positions where no such rocks are visible, a notable instance of this being at Melton-Mowbray, where there are no igneous rocks in the immediate neighbourhood, although the needle is deflected through as much as 67°, which may be due to the presence of large masses of basaltic rock lying beneath the surface.

Prof. A. W. Rücker,* speaking of this source of attraction, says:—"Magnetic rocks are often permanently magnetized, and may deflect a compass held close to them through 40° or 50°. The permanent magnetization is, however, irregular, and at a comparatively short distance the disturbing effect appears to be due almost exclusively to the uniform magnetization produced by the earth's magnetic field."

The general conclusions which he comes to, both as regards theory and experiment, are:—(*a*) "That dykes and thin uniform basaltic sheets produce no measurable effects except at distances from their edges, which are small multiples of their thickness; (*b*) That isolated masses of trap-rock, a few square miles in area, produce no important magnetic effects at distances comparable with their linear dimensions."

Induction Currents.—Induction currents may be produced by the neighbourhood of currents or of magnetic bodies inducing them, *i.e.* rocks containing iron or its compounds.

* *Trans. Inst. Min. Eng.*, vol. ix. p. 420.

Magnetic Storms.—The irregular fluctuations of the needle which occur from time to time proceed from magnetic storms. The word storm in this connection does not necessarily imply violent action, although occasionally the needle oscillates backwards and forwards. As a general rule, however, the needle becomes temporarily deflected some degrees beyond the normal variation over a large area, sometimes for 24 or 48 hours. Magnetic storms are invariably connected with displays of aurora, and, in some unexplained manner, with the occurrence of spots on the sun, and, as shown by Prof. Balfour Stewart, they commonly occur at intervals of ten or eleven years, when the sun spots are at their maxima. If the compass is used during one of these storms the needle sometimes becomes practically worthless for the time being.

Influence of Rails on the Needle.—The presence of iron or steel may cause the needle to deflect. Cast-iron in small quantities does not seem to have much influence, and those used underground at many Scotch collieries (weighing 28 lbs. per yd.) have so little effect that the surveys of those mines are almost invariably made with a free needle. Malleable iron rails have a more appreciable effect. With rails weighing 12 to 15 lbs. per yd., the deflection is seldom greater than 2° when the compass is placed *in the centre of* the two rails, and about 3 ft. above the ground.

Steel rails of the same weight exert more influence, and the needle is often deflected over as much as 10° to 15° in the presence of such rails, so that special precautions become necessary in making observations. Steel ropes and tools will also affect the needle, and in order to get a true reading the compass should be planted at least 15 ft. from any metal, while the presence of a large mass of metal necessitates its being placed still further away. In highly inclined workings, with the dip towards the north, it frequently happens that if a road is driven in the direction of the magnetic meridian, the rails become strongly magnetized on account of the earth currents passing longitudinally through them. Rails in such a position influence the needle to a much greater extent than would otherwise be the case.

Surveying in the presence of iron may be carried out with approximate accuracy if readings are taken at every station instead of at every second station, and by taking back and foresights. In these circumstances, both back and foresight readings will be subject to the same degree of error, and, consequently, if a correct bearing of any particular line can be obtained, a fairly accurate survey can be made.

Dip of the Needle.—A magnetic needle does not, in our latitudes, assume a perfectly horizontal position. The inclination of a freely suspended needle is about 67° at Glasgow towards the north. The dip varies in the same way as the declination, and ranges from 75°, the maximum, to 66°, the recorded minimum at Greenwich. In mining dials this tendency is counterbalanced, so that the needle moves only

in a horizontal plane. All needles have not exactly the same variation, and for this reason it is very important that magnetic bearings made at the surface should be taken with the same compass that has been used for the underground surveys.

The length of needle in common use for underground work is 5½ in. It consists of a strip of steel with an agate or ruby centre, and a cross-cut line marking the north-seeking end. In some cases a small vernier is fixed to one end, so that the bearing may be read to the nearest minute.

The dial plate is usually constructed of brass, although in some instances aluminium has been used with success. The figures may be marked on this plate in various ways, according to the practice of different makers. The method of reading will be explained later.

Determination of True Meridian.—To determine the true meridian, and from this the exact amount of magnetic declination, various methods may be employed, the best two being to make observations of the Pole star or of the sun.

Observations of Pole Star and Sun.—This method is accomplished by using a theodolite and bringing the telescope of the instrument to bear upon the Pole star at stated intervals before and after its culmination. A reading is taken several hours before its culmination, both on the horizontal and vertical circles of the theodolite. After a lapse of the same period subsequent to its culmination the star is sighted a second time, and a reading taken as before. The line bisecting the angle obtained by the two readings will indicate the true meridian. If the observations are taken during winter, which is the best time, the readings are taken at an interval of 11 hours 58 minutes for the Pole star. Similar methods are resorted to of finding the meridian by means of observations of the sun.

Setting out the Meridian Line.—In every mining district a permanent meridian line should be set out, so as to enable surveyors to determine the true meridian at any time.

The best method of permanently marking out the meridian line is to insert in the ground, 4 or 5 ft. deep, a large hard stone, granite if possible, 6 or 8 ft. long × 2 ft. broad. The stone should be well faced and firmly set in cement to keep it from shifting. In the upper surface of the stone is fastened a brass plate, a foot square, let in so as to be perfectly horizontal, and on this the meridian line is shown by a fine engraved line. For practical purposes a point may be fixed at some part of the royalty, and a line laid out connecting it with some permanent point, such as the centre of the shaft, and the bearing of the magnetic with this line recorded every year or every second year; the difference in the readings will be the amount of the annual variation.*

* *A Treatise on Mine Surveying*, p. 50.

A very convenient method is to fix a pin in the ground on some part of the colliery which is unlikely to be affected by the proximity to underground workings, and to sight from this point a number of permanent objects within range, such as steeples, factory chimneys, etc. The mean of the angles made by the axis of the magnetic, and the imaginary lines connecting the point with the objects selected, determines the variation in the meridian in a more satisfactory manner than by relying on observations in one line only.

To ascertain if there are any local attractions affecting the needle, a number of observations should be made on a straight line, the correct bearing of which is known.

Colliery Plans.—A number of plans are published by the Ordnance Survey Department which are of great use in mining operations. The smallest are on a scale of 1 in. to the mile, showing the roads, railways, and chief land marks, such as farmhouses, plantations, streams, etc. The next size is 6 in. to the mile, showing every detail, such as fences and other boundaries. On these plans the levels are also marked, showing the height of various points above the mean sea-level at Liverpool. The positions at which the levels have been taken are indicated by dots, and the height marked beside them in figures, thus ⊙ 683·3. Where these observations have been made, *bench marks*, as they are called, are cut on the walls of buildings, pavements, etc. Their positions are indicated thus:

Contour lines are also indicated on these plans by means of dotted lines. These contour lines indicate that each point on the line is at the same height (given in small figures within the areas thus outlined) above sea-level.

This information is very useful to the mining engineer in aiding him to determine the levels of the various pits and bore-holes which may be put down on an estate, and in enabling him to arrive at a conclusion as to the best position in which to sink new shafts or to lay down sidings and other works. The 1 in. and the 6 in. maps can be obtained coloured, showing the geological formation of the various coalfields and the outcrops of the various seams, the depths of the pits sunk, the position and extent of faults, where proved, and much other useful information. The largest scale plans published by the Ordnance Survey is 25·344 in. to the mile, or $\frac{1}{2500}$ of the actual size of the district surveyed. These plans are very accurate, and show every surface-feature that was in existence at the time of the survey, together with the bench marks, etc. Unfortunately, the paper on which the maps are printed is found to contract and expand considerably, and this may sometimes lead to an error, but that may be guarded against to a certain extent by using the scale printed on the plan itself in preference to a detached scale. On these sheets every enclosure is numbered, and a book of reference to these numbers is published for each parish represented, giving the

area of these enclosures and the nature of the ground, such as pasture land, wood land, gardens, etc., etc. In the latest editions of these plans the area of each enclosure is marked.

The levels marked on an Ordnance Survey plan are not always correct for colliery districts, as the continual subsidence of the underground workings affects the surface level. As an instance of this it was found recently, when a new survey of one of the Scotch coalfields was made, that nearly every bench mark had subsided from 5 to 7 ft.

The Ordnance Survey also publish vertical sections of the various coalfields, corresponding with lines shown in the plans, showing the seams, and horizontal sections showing the nature of the intervening strata.

The smallest scale permitted for colliery plans, by the provisions of the Coal Mines Regulation Act, is $\frac{1}{2500}$. For the working plans of collieries this scale is, however, too small, and the scale in common use is $\frac{1}{2}$ in. to 66 ft. Sometimes scales of 1 in. to 66 ft. are used, but a plan of an extensive royalty drawn to that scale becomes inconvenient and unwieldy to work from. In preparing a colliery plan the first step is to make an accurate survey of the surface boundaries and determine the position of the various shafts and surface buildings. This is usually done with a theodolite. The details of fences, etc., may be filled in by enlargement from the 25 in. Ordnance Survey maps, but as many of these plans are very much out of date, it is often necessary to make a complete new survey of all the surface lines within the boundary. For this purpose the theodolite may be used, but where the details are somewhat intricate it is more expeditious to use the mining compass. A plan which has merely been enlarged from the 25 in. scale without check surveys cannot be relied upon when workings are being carried on near the boundaries, but it may be useful as a general plan to guide operations.

The information which should be shown on colliery plans is as follows:—All underground workings showing the position of coal faces and the position of all pillars of coal; faults and dykes showing the direction and amount of throw; all the air and water-courses and the haulage roads. The thickness of the coal should be marked at frequent intervals, and special care should be taken to indicate clearly what has been found in exploration workings. The dip and rise of the strata should be indicated with an arrow and the rate of inclination marked thereon. Levels should also be marked at various points, showing the height above a given datum line, one, two, or three thousand feet below sea-level being convenient datum lines.

The date of the survey of all working faces should be neatly marked and the workings should be surveyed and extended in ink every three months. The position of all shafts should be shown and the depth marked, and a section must be given showing the strata

sunk through, or, if that is not practicable, one showing the average thickness of the coal and the nature of the roof and pavement. The surface lines are usually drawn in with Indian ink, and the boundaries indicated by an edging of colour.

To indicate the underground workings, colour is generally used instead of Indian ink, a separate colour being used for each seam. Faults and troubles should be shown distinctively coloured. When two or more superimposed seams are worked, and require to be shown on the same plan, each seam is coloured differently, as stated above, but it is far better to make a separate plan for each seam worked. In pillar and stall workings only the road need be coloured, while the solid is being worked, but after the pillars have been removed it is usual to colour the waste or exhausted area and show it by hatched lines.

In longwall workings the waste only need be coloured and the roads left uncoloured. Roadways going through solid coal are generally shown by unbroken lines, while those going through waste, as in longwall, are usually shown with dotted lines. The two commonest colours used in making plans are red and blue. On surface plans water is shown blue, roads 'Burnt Sienna,' and buildings indicated in a wash of Indian ink. It is generally found sufficient to colour the roads, buildings, and water.

All colliery plans should be drawn on strong mounted paper, and great care should be taken that the paper is thoroughly seasoned before being used.

Underground Surveying.—For the great majority of underground surveys the instrument used is some form of mining compass. In cases where there is no disturbing magnetic influence, such as in surveying the roadway shown in fig. 488, the compass is first planted at B and then carefully levelled, the needle being allowed to settle. A lamp is now held at A in the centre of the shaft, or on one of the guides, and the sights of the compass rotated until the hair cuts the light. The bearing is then read off and recorded in the survey book, the measurement being taken at the same time by the assistants, and likewise recorded in the book. By this time a man has gone forward to the point C, and places there a lamp on the floor as near to the centre of the road as possible.

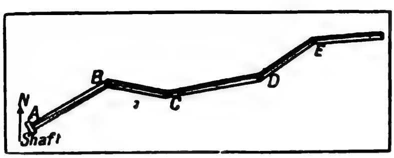


FIG. 488.

The bearing from B to C is now taken and recorded in the same way. The surveyor then lifts the compass and plants it at D, and takes the bearings D C and D E in the same way as before. There are various ways of booking a survey like this, two of the commonest methods being here shown.

First Method:

No. of Bearing.	Bearing.	Distance Links.	Remarks.
(1)	N. 70 E.	100	From centre of No. 1 Pit.
(2)	S. 85 E.	126	
(3)	N. 75 E.	167	
(4)	N. 5 E.	122	

Second Method:—With the second method it is desirable to make a small sketch at the same time as the bearings are being written down, so as to give the surveyor a general idea of the ground he is surveying (fig. 489).

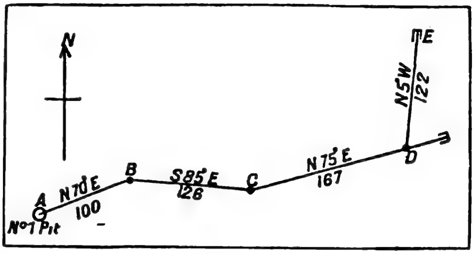


Fig. 489.

This is a very satisfactory and expeditious method of surveying, provided the surveyor has had sufficient practice to sketch neatly. There are one or two points which need to be taken note of. Take bearing No. 2, which reads S. 85 E. A common mistake among learners in marking a bearing like this would be to record it as N. 85 E., and a mistake like that is almost impossible to check when the survey comes to be plotted. Take now the bearing B C, as shown on fig. 490; whichever end of the needle may be looked at there is only the one figure which can be read off, viz., 85°, and this is accordingly marked in the book. Similarly the last bearing might have been marked N. 5 W., whereas in the other case it is simply marked 175°, and it is hardly possible to make a mistake. The only mistake which is liable to occur with this latter method is that such a bearing as D E might have been sketched as turning to the right instead of to the left. A mistake like that, however, is easily detected when plotting, and with a little care underground in watching the general direction of the road, whether towards the north or towards the south, this source of error should be easily avoided.

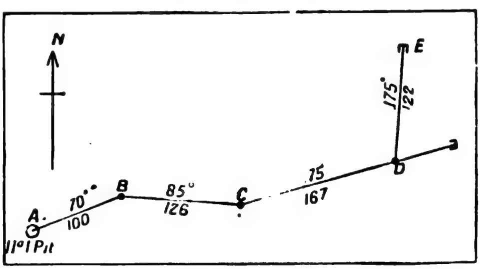


Fig. 490.

It occasionally happens that a few bearings have to be taken at

some point where magnetic attractions are exerted, and that a free vernier compass cannot be employed for the purpose. To do this with the loose needle the method usually adopted is shown in fig. 491.

Planting the compass at B, the bearing reads 60° from B to A and 90° from B to C. Planting again at C, a back sight is taken to B and reads 83°, and a foresight to D reads 135°. Similarly at D the back sight reads 133° and the foresight 70°, and at E the back sight is 73° and the foresight is 92°. At E there is no iron to attract the needle, but as a check back sights are taken at F, and as this also records 92° it indicates that the bearing taken from E is correct. Now the bearing from E to D is 73°, and this bearing having been taken from a point where there is no attraction is bound to be correct. When the compass was at D the bearing from D to E read 70°, therefore the amount of attraction at D is 73 − 70 = 3°, to be added to the observed bearing, so that the bearing 133 taken from D to C is 3° wrong and should be 136°. In the same way at C the correct bearing C to D is 136° and the compass reads 135°,—that is, 1° to be added to the compass bearing, so that the correct bearing C to B is 83 + 1 = 84°. Again, at B the compass reads 90 from B to C instead of 84°, so that 6° must be *deducted* this time from all bearings taken at the point B, and the correct bearing from B to A becomes 54°.

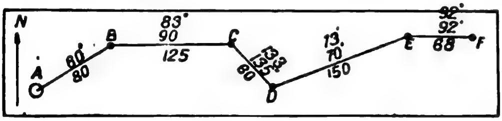


FIG. 491.

The survey would now be plotted :

A to B 54°, B to C 84°, C to D 136°, D to E 73°, and E to F 92°.

The student should never think of seeking the *average* bearing of the back and foresights in a survey of this kind, as such a method would be quite incorrect, and a careful surveyor would endeavour to secure a sight free from magnetic disturbance at each end of the survey, so that the one may be checked by the other.

The Fixed Needle or Vernier Compass.—This instrument may be used in two ways underground. The first method is by taking the angles between each two bearing lines by placing *the vernier at zero* at every station, as is sometimes done with the theodolite when taking a surface survey. This method of surveying is, in the writer's opinion, quite unsuitable for general underground work, and will not be further described.

The second method is to take the bearings and book them precisely the same as if they were loose needle bearings.

Let fig. 492 represent a working which is to be surveyed with the vernier along the road from A to F, which is laid with steel rails. Suppose that at B[1] there is an old roadway without rails, and where the compass can be planted at a sufficient distance back from the

main road to avoid disturbing influences. The first operation is to set up the compass at B^1. Fix the arrow on the vernier plate opposite zero, then unclamp the needle, and after allowing it to settle turn the dial round until the needle also points to zero.

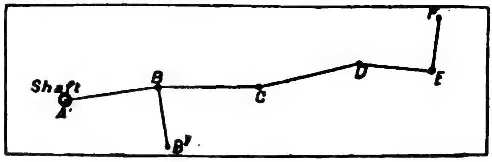


FIG. 492.

The clamp screws are then tightened to secure the compass to the tripod and the clamping screw of the vernier slackened. If the dial is now turned round, the arrow on the vernier plate still remains pointing to the north, as it will be moving along with the north end of the needle. The sights are now turned to cut the light held at B and the vernier screw is then tightened up. Before lifting the compass a lamp is placed vertically below the centre, and for this purpose a brass plumb line with a thin flexible cord is used. The instrument is now lifted and planted exactly above the lamp at B, and carefully levelled. The lower screws of the tripod are loosened and the compass turned to sight back to the lamp at B^1. When this has been done the lower screws are again tightened up. It is evident now that the compass, as planted at B, is lying in exactly the same line as it was when placed at B^1 and the arrow on the vernier plate still points to north. The vernier clamp screw is now slackened and the bearing B to A taken and recorded in the survey book. The bearing B to C is also taken, and while looking through the compass to C the vernier clamp screw is tightened. The compass is again lifted forward to C, and before the vernier is slackened the back sight towards B is taken, so that the vernier still remains pointing to the north. The bearing C D is now taken. In the same way the bearings D E and E F are also taken. Suppose F is at the coal face, and that it is possible to obtain a clear bearing at that point. The vernier may now be checked to see if it still corresponds with the needle. If it does the survey may be taken as correct, but the student will very often find that his work has not been done with sufficient accuracy, and that the needle does not coincide with the vernier. If the work is important a new survey must be made, but in many cases it will be sufficient to make an allowance for the error when plotting the survey. For instance, if the last bearing E F in fig. 492 is 1° wrong, then this error may be divided among the total number of bearings (which in this case is five), so that 12 minutes would require to be added or deducted, as the case may be, in each case. It is not always possible to get a clear bearing at the commencement of the survey. In such a case the survey may be taken from the face outwards, or if that is not convenient then a start may be made at the pit

bottom the vernier being put at the incorrect north indicated by the magnetic needle, and the survey made inwards in the usual way until a clear bearing can be obtained. If at this point the needle is, say, 5° different from the vernier, then this difference must be allowed on each bearing up to the point where the clear bearing was obtained. The vernier is then adjusted to coincide with the magnetic needle, and the survey proceeds as before. When a side road, off the main road, has to be surveyed with the vernier it is necessary to leave in two marks on the main line of survey. Suppose at B, on fig. 493, the road B D E F has to be surveyed. The bearing A B is taken and a mark left at A, the bearing B D is also taken and a lamp planted below the compass at B. The instrument is now shifted forward to D, a back sight taken, and the position of the lamp at B is now carefully marked and the bearing D E taken, and so on, till F is reached. The compass is now brought back and carefully plumbed over the mark left at B and the vernier fixed at the bearing A B, as before obtained. The back sight is now taken to A, the vernier unclamped and the foresight taken to C, the survey proceeding in the usual way. To insure accuracy when working with the vernier a good plan is to use two tripods. The spare tripod is planted at the point where the foresight is to be taken, and a lamp is held exactly at the centre. After taking the sight the compass is lifted off its tripod and carried forward and placed on the spare tripod, and the back sight taken to a light held exactly at the centre of the last tripod. The surveyor has then to wait until this tripod is brought forward and planted in the position for the front sight. The learner working with the vernier often makes mistakes by slackening the wrong screws, with the result that the whole work has to be done over again. Assistants, too, often lift the lamp from the back sight point before it has been looked at, with the result that the base line for the forward sight is destroyed, and part of the work has to be repeated. These mistakes should be carefully avoided.

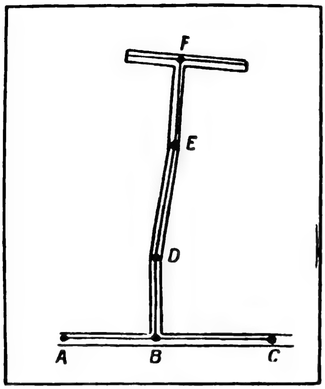


FIG. 493.

Measurements.—In surveying, lengths are usually measured by the imperial chain, which consists of 100 links, each link being 7·92 in. long, and therefore its total length is 66 ft. By adopting this unit of length, the chain is made to bear certain ratios to other standard measurements; for instance, $\frac{1}{4}$ chain is equal to 1 pole, and 1 mile equals 80 chains, and 10 chains = 1 furlong. Again, 1 acre =

4840 sq. yds., and $22^2 = 484$ yds. = 1 sq. chain, therefore we have 10 sq. chains in 1 acre, and as 10 sq. chains are equal to 10000 sq. links, 100,000 sq. links are equal to 1 acre.

Measurements are also made by chains 100 ft. in length; these chains are largely used in civil engineering work, and also in metal mining districts. By using such a length of chain, measurements can be made much more rapidly, and are easier dealt with for surface surveys. The unit of measurement in ore mines is usually the imperial chain, but very often the unit is 1 fathom and a chain used 10 fathoms long. On the Continent the unit is 1 metre, which is equal to 39·37 in.

The best chains are made of the finest mild steel, as with this material they can be made light, and less easily bend.

There are many sources of inaccuracy which arise from using the chain, and great care has to be taken to avoid them. The chain should be frequently checked by some standard mark, as it is very liable to become stretched, or some of the links may be bent when it becomes too short.

In most large towns standard measurements are marked with brass plates on some public building. The chain should be checked at one of those standard marks, and similar permanent marks should be laid out at the colliery, so that the chain may be checked at any time and always kept correct.

Care must also be taken when measuring that none of the links are interlocked, as this would also cause considerable error.

Another method of measuring is by means of steel tapes. These tapes are largely used in some American mines. In Pennsylvania, tapes 300 to 600 ft. long are employed. Measurements taken in this way are likely to be more accurate than with a chain, but steel tapes must not be allowed to kink, as they are then very apt to break. The difficulty connected with tapes is the reading of them in muddy and wet mines, where they get rusted. To obviate this, small bits of brass wire, with a certain number of nicks or cuts in them, are soldered on at regular distances apart.

For very correct measurements, surveyor's rods are used; these are made of lancewood, in pieces about 5 ft. long, joined together by a 'scarf' joint. They are often used to check the length of the other measuring instruments, and are also useful for marking out a standard chain. Glass rods have also been used on important government surveys.

Measuring on the Slope.—In chain work on the slope, corrections must be made to get the proper horizontal measurement. There are several methods of doing this, such as: (1) by ascertaining the angle of inclination; (2) measuring along the slope and making deduction from every measurement; (3) by 'stepping' the chain. When incline measurements are required to be accurately made, the angle of inclination is ascertained by a clinometer or similar instrument,

and the actual horizontal measurements can be ascertained by calculation from tables compiled for the purpose. Many mining compasses are fitted with an external arc for taking the angles of inclination simultaneously with the horizontal angles.

The method that is most largely adopted by surveyors is to measure along the slope and make deductions for each length, according to the inclination, the amount of deduction necessary being derived from the tables. In the absence of these, the following approximate method may be employed.

Example.—A seam dips 1 in 8; find the horizontal lengths corresponding to the following measurements, 360, 470, 832.

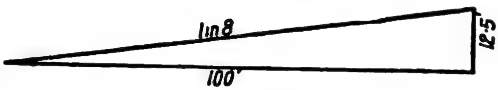


FIG. 494.

If L is the horizontal length required, D the measured distance on the slope, and d the dip, $L=\sqrt{D^2-\left(\frac{D}{d}\right)^2}$

$$\therefore (1)\ L=\sqrt{129600-2025}=357;$$
$$(2)\ L=\sqrt{220900-3361}=465; \text{ and}$$
$$(3)\ L=\sqrt{692224-10816}=825.$$

In the above examples decimal fractions have been disregarded, although in practice they must be considered, or the result may lead to error when considerable distances are involved.

Stepping the Chain.—This is a method sometimes adopted to ascertain the horizontal measurement on inclined workings, and is very expeditious, but only approximate results can be obtained, and the method should not be adopted on gradients of more than 5°; when the inclination is greater it is too inaccurate.

The method of 'stepping' the chain will be understood from fig. 495. The chain should be taken in not more than half lengths or quarter lengths, and stretched out firmly to A, say, and a cord and plumb-bob dropped vertically to B, the distance O A being then carefully measured. The same process is repeated at C and E, the distance B C and D E being found in the same way. The vertical distances A B, C D, E F need not be measured, unless they are wanted as a check on the work.

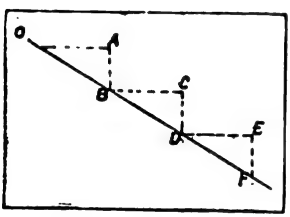


FIG. 495.

Plotting the Survey.—The lines and angles of a survey are usually represented on paper on a very small scale compared to the original lengths and angles. The process of

putting these lines and angles on a plan is termed plotting. The plotting of surveys may be divided into three different methods, viz., by means of protractors, by rectangular co-ordinates, and by chords. The first method is the one most largely used for colliery work, and can be done very accurately with a good protractor.

Protractors.—There are various kinds of protractors used for this purpose, the commonest sort being a half circle with or without a movable arm.* They may be made of brass, ivory, celluloid, or white metal. The semicircle is divided from 0° to 180° in opposite directions. To plot the survey the straight side of the protractor is laid along an assumed meridian line (fig. 496), and the bearings pricked off by making a mark with a fine pencil or needle at the desired angle and numbering the bearings as they are taken off. All bearings say to south-east and north-east will be first taken off and

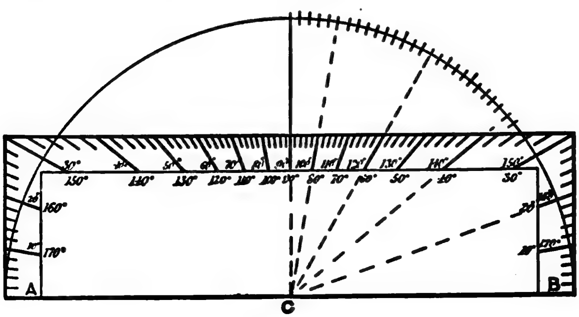


Fig. 496.

numbered; the protractor will then be reversed and all the bearings to south-west and north-west pricked off and likewise numbered. In laying the protractor on the meridian line its centre should be placed on some fixed point, such as the intersection of a line drawn at right angles to the meridian line, which would, of course, represent an east and west line. The first line may then be taken from this centre point to the first bearing and the distance marked off, or it is sometimes more convenient to parallel the line to a different part of the paper, clear of the bearings altogether. For this purpose two set-squares or a parallel ruler may be used, but the latter should not be used unless it is of fair size and provided with rollers. The disadvantage of this method of plotting is that the plan gets spoiled by the large number of pencil or needle marks that would be required. For office work the semicircular protractor is often provided with a movable arm and a small vernier, which enables it to be used with

* The protractor shown in the figure is made of boxwood. A more reliable instrument is a semicircular protractor made of horn.

great accuracy. A long steel 'straight-edge' is laid along the meridian line, and the protractor is fitted on to this with its straight side against the straight-edge. The bearings and lines can then be taken off rapidly and accurately by means of the movable arm, and without requiring to prick them off or mark them on the plan.

The circular protractor divided to the full circle is more accurate and convenient for plotting a survey. It may be used in much the same way as the semicircle. When provided with a swinging arm the bearings can be taken off very rapidly and accurately. Another method of plotting is to use a large cardboard protractor and parallel ruler. This cardboard protractor is usually 12 to 18 in. square and divided from 0° to 360°, numbered in opposite directions. The centre portion of the cardboard is cut out, and the north and south line made to coincide with the plan meridian. The parallel ruler is placed on the required bearing and transferred direct to the plan. This saves much time and keeps the plan free of pencil marks, which are necessary with the ordinary circular protractor. The bearings can be taken very accurately, owing to the large diameter of the protractor. In the making of the Government Ordnance Survey Plans such protractors are always used.

In Scotland a method of plotting, which can be very rapidly and accurately done by what is known as a 'table' protractor, is largely used in mining engineers' offices. This protractor consists of a large circle 2 ft. in diameter. This circle is fitted into a square wooden frame or table *a*, in which it revolves on a central axis *b*. On the inside revolving circle is fixed a circular sheet of paper or scroll plan, with the working meridian on it; this line is made to coincide with the north and south line on the protractor. A small brass plate *c*, with an arrow engraved on it, is fixed to the square frame, and the bearing required is made to coincide with this arrow, the line being transferred on to the paper direct by the aid of a T-square laid across the protractor. The construction and method of using this protractor will be understood from the illustration in fig. 497.

The bearings, after being plotted on these scroll plans, are transferred to the large colliery plans by tracing paper. A large amount of work can be done very rapidly by this method, and the large size of the protractor insures its accuracy.

Plotting by Co-ordinates.—Where surveys have to be plotted with great accuracy, the method of using rectangular co-ordinates is to be preferred to any other. * It consists in assuming two fixed *axes* crossing at right angles to each other at a fixed point or origin, and in calculating the perpendicular distances, or *co-ordinates*, of each station from these axes. If the true meridian has been ascertained, it may be made to represent one of the axes. In fig. 498, which illustrates this method of plotting, the north and south line, and east and west drawn at right angles have been taken to represent the required axes.

* *A Treatise on Mine Surveying*, by Bennett H. Brough, pp. 140–143.

Now every line that is taken in a survey will depart from the north and south line by a definite amount, according to its angle and distance, and similarly it will also depart a certain amount from the east and west line. The distance that the line departs from the north and south line is termed its *latitude*, and the distance that it departs from the east and west line is termed its *departure*. The latitude of a point may be defined as its distances due north or south of some fixed point. The distance that one end of a line is due north or south of the other end is called the difference of latitude of the

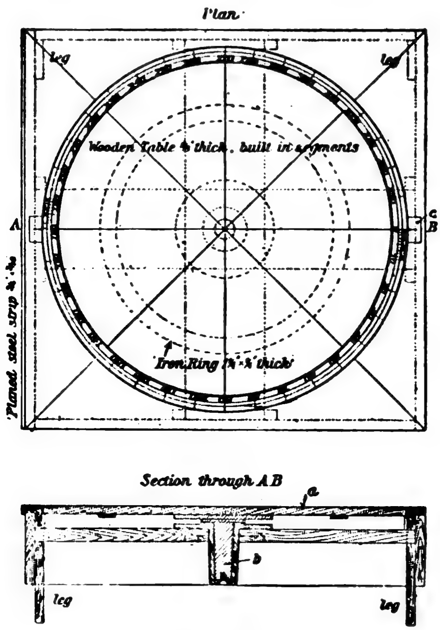


FIG. 497.

ends of the line, and similarly the distance which one end of a line is east or west of the other end is called the difference of longitude of the ends of the line or the departure.

The latitude = distance × cosine of bearing.
" departure = distance × sine of bearing.

To obviate the tedious process of calculating each bearing, the latitudes and departures may be taken from a book of traverse tables. The best book of tables for this purpose is that by R. Lloyd Gurden,

in which the tables are computed for every minute of angle. The data for the survey plotted in fig. 498 were as follows:

Bearing.	Distance.	Latitude.		Departure.	
	Links.	N +	S −	E +	W −
N. 10° E.	100	98·5	...	17·4	...
N. 77° E.	130	29·2	...	126·7	...
S. 67° E.	134	...	52·4	123·3	...
S. 53° E.	136	...	81·8	108·6	...
S. 20° W.	42	...	39·5	...	14·4
S. 29° E.	132	...	115·4	64·0	...
S. 42° W.	86	...	63·9	...	57·5
S. 89° W.	78	...	1·4	...	78·0
N. 70° W.	110	37·6	...	...	103·4
N. 65° W.	106	44·8	...	...	96·1
N. 85° W.	47	4·1	...	...	46·8
N. 66° W.	120	48·8	...	...	109·6
N. 53° E.	113	91·4	...	65·8	...

Before these co-ordinates can be used for plotting, they must be reduced to total latitudes and departures. To effect this the northings and eastings are regarded as positive quantities, while the southings and westings are regarded as negative quantities, and the reductions are then computed by taking the algebraical sum at each station, which from the above data would be as follows:

No.	Total Latitude.	Total Departure.
(1)	+ 98·5	+ 17·4
(2)	+127·7	+144·1
(3)	+ 75·3	+267·4
(4)	− 6·5	+376·0
(5)	− 46·0	+361·6
(6)	−161·4	+425·6
(7)	−225·3	+368·1
(8)	−226·7	+290·1
(9)	−189·1	+186·7
(10)	−144·3	+ 90·6
(11)	−140·2	+ 43·8
(12)	− 91·4	− 65·8
(13)	00·0	00·0

To plot from this table, draw a north and south line, and an east and west line, making the junction of these two lines the starting point for the first station. From this point (A, fig. 498) take line No. 1 and measure off 98·5 northwards and 17·4 eastwards, making a mark at the end of these distances. From the end of distance 98·5 draw a line at right angles to north and south, and from the end of distance 17·4 draw another line at right angles to east and west, continuing this line till it meets the line drawn at right angles to north and south. Join the meeting point of these two lines to the point A, and this will give the first line of the survey. The same procedure is followed in regard to the other lines, measuring off all the positive latitudes to the north and negative latitudes to the south,

while the positive departures go to the east and negative departures to the west. The student will understand this method more fully if he plots down the above survey once or twice, and works out the numbers for himself. If care be taken when getting out the ordi-

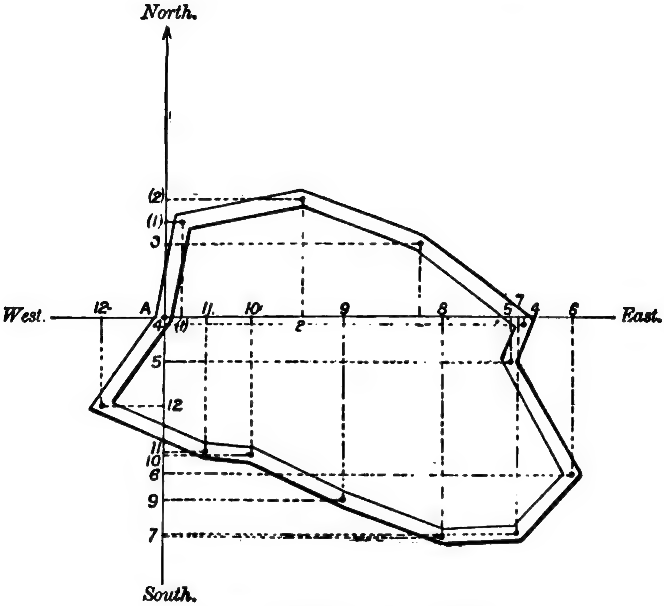


FIG. 498.—Plotting by Co-ordinates.

nates, this method of plotting insures great accuracy, and is specially useful for important surface or other surveys.

Plotting by Chords.—A tied survey may be plotted from one meridian, without the aid of a protractor, by using a table of chords

No.	Bearing.	Chords, Radius = 1000.	Distance.
(1)	N. 10° E.	174	100
(2)	N. 77° E.	1245	130
(3)	S. 67° E.	1103	134
(4)	S. 53° E.	892	136
(5)	S. 20° W.	347	42
(6)	S. 29° E.	500	132
(7)	S. 42° W.	716	86
(8)	S. 89° W.	1401	78
(9)	N. 70° W.	1147	110
(10)	N. 65° W.	1074	106
(11)	N. 85° W.	1351	47
(12)	N. 66° W.	1089	120
(13)	N. 54° E.	892	113

In the survey already shown plotted by co-ordinates, using the same data, and from the table of chords given, we would have to draw a circle on the paper, with a radius preferably of 10 in., and through the centre of the circle draw a north and south line and an east and west line. With a pair of dividers and a scale, lay off the chords along the circumference of the circle according to the direction of the bearing. The chords being taken from a radius of 1000 would have to be reduced by two decimal places to plot with a circle whose radius is 10, thus the first chord would be 1·74, the second 12·45, and so on. When the chords are all measured off, the rest of the work is performed in the same way as when a circular protractor has been used. This method of plotting is not to be recommended for colliery surveying, as drawing circles repeatedly on the plan and pricking off the chords would soon break the surface of the paper and spoil it.

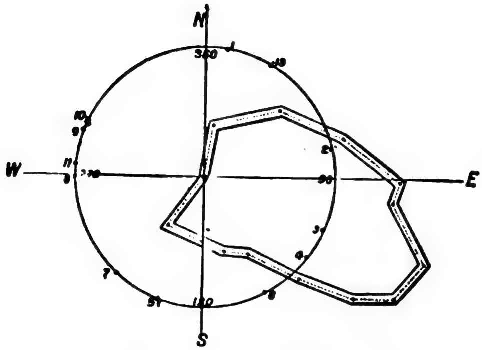


FIG. 499.—Plotting by Chords.

Calculation of Areas.—The three principal methods of calculating plan areas are: (1) by dividing the total area into triangles or squares; (2) calculating the area by means of ordinates; (3) by mechanical methods, such as measuring instruments. The areas of coalfields are nearly all calculated in acres, roods, and poles. The statute acre is equal to 10 sq. chains or 100,000 sq. links.

Calculation by Triangles.—The plan area may be divided into a number of triangles, either right angled or otherwise. If the triangles are right angled ones the calculations are much simplified. When the triangle possesses a right angle (fig. 500) the area

$$A = \frac{\text{base} \times pp}{2}.$$

If none of the angles are right angles, as in fig. 501, then an angle of 90° may be constructed by dropping a perpendicular pp from any apex on to the opposite side, and the area of the two triangles thus constructed may then be found separately. Or if we have two sides and know the included angle, as in fig. 502,

$$\text{then the area } A = \frac{ab \times \sin \alpha}{2}.$$

When none of the angles are known the following formula is used for calculating the area,

$$A = \sqrt{S(s-a)(s-b)(s-c)}$$

where S is the semi-perimeter or half the sum of the sides. If logarithms are used, the formula becomes

$$\text{Log } A = \frac{1}{2}\left\{\log s + \log(s-a) + \log(s-b) + \log(s-c)\right\}$$

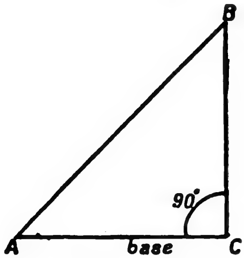


FIG. 500.

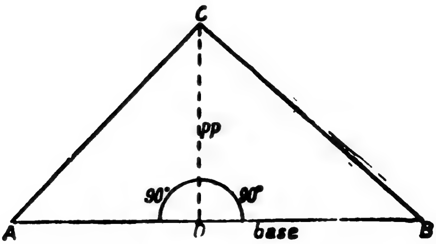


FIG. 501.

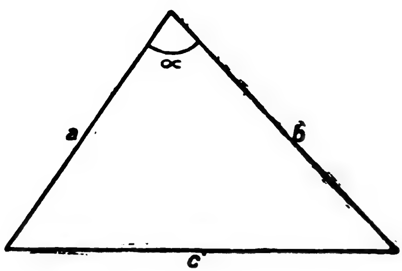


FIG. 502.

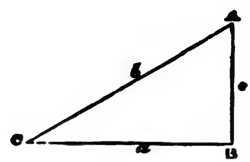


FIG. 503.

In the above triangle, if the respective sides a, b, and c are 1200, 1100, and 1000 links, find the area in acres, roods, and poles.

$$\text{Here } S = \frac{1200 + 1100 + 1000}{2} = 1650$$

$$\therefore A = \sqrt{1650\{(1650-1200)(1650-1100)(1650-1000)\}}$$

$$= \sqrt{1650 \times 450 \times 550 \times 650}$$

$$= \sqrt{265443750000}$$

$$= 515209{\cdot}37 \text{ sq. links.}$$

To reduce this to acres, roods, and poles, count off five decimal places thus :—

$$
\begin{array}{r}
5{\cdot}15309 \\
4 \\
\hline
0{\cdot}61236 \\
40 \\
\hline
24{\cdot}49440 \\
\hline
\end{array}
\quad \therefore \text{ Area} = 5 \text{ acres, } 0 \text{ roods, } 24{\cdot}49 \text{ poles.}
$$

Or by logarithms

$$\text{Log } A = \tfrac{1}{2}\{\log 1650 + \log(1650 - 1200) + \log(1650 - 1100) + \log(1650 - 1000)\}$$

$$= \tfrac{1}{2}\{3{\cdot}2174 + 2{\cdot}6532 + 2{\cdot}7408 + 2{\cdot}8129\}$$

and Log $A = 5{\cdot}7119$, and $A = 515310{\cdot}00$ sq. links.

Calculation by Ordinates.—When the area to be calculated has an uneven outline, and is not of too great extent, the method of ordinates may be used. It consists in running a chain line or axis through the greatest length of the area to be calculated, and taking offsets at right angles to the chain line. The offsets should be taken sufficiently close so as to make each figure approximately a trapezoid, therefore the more offsets are taken the more accurate will the calculation become.

An example will show more clearly this method of calculating areas.

	B	
150	1550	0
182	1300	...
...	1248	175
...	1159	55
...	980	183
280	865	...
202	393	92
...	150	75
145	45	...
0		0
	A	

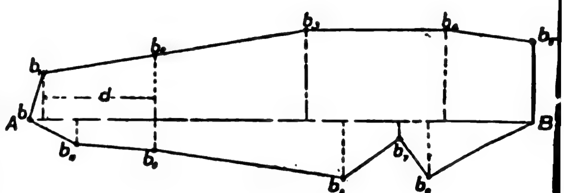


FIG. 504.

From the above data an irregular field was surveyed by running a chain line through it from A to B, and taking offsets right and left. By referring to fig. 504 it will be seen that the first offset on the left hand b is o, and the next b_1, 45 links along the chain line is 145, so that if we call the distance between the two offsets d, the area of this trapezoid $= d\left(\frac{b+b_1}{2}\right) = 45\left(\frac{0+145}{2}\right) = 3262{\cdot}5$ sq. links, and so on with the others, taking b_1 and b_2 for the next trapezoid. We may avoid the division by 2 for each trapezoid, by dividing the total sum of all the trapezoids at the end. Taking the offsets on the left hand

side first from A to B and the right hand from B to A, we get the following :—

(1)	45	(0+145)=	6525
(2)	348	(145+202)=	120756
(3)	472	(202+280)=	227504
(4)	435	(280+182)=	200970
(5)	250	(182+160)=	83000
(6)	302	(0+175)=	52850
(7)	89	(175+ 55)=	20470
(8)	179	(55+183)=	42602
(9)	587	(183+ 92)=	161425
(10)	243	(92+ 75)=	40581
(11)	150	(75+ 0)=	11250

967933 ÷ 2 = 483966·5 sq. links.

The area would therefore be 4 acres, 3 roods, 14·34 poles.

If the area to be calculated is large this process would be tedious, and areas with curved outlines can often be calculated with sufficient accuracy by the process of *equalising* or *giving* and *taking*; *i.e.* to draw a straight line through the curved boundary, leaving as much space outside the straight line as there is inside it. The whole area is then divided into triangles and calculated as already shown.

Calculation by Instruments.—This method is now much resorted to in mining engineers' offices for calculation of plan areas. The instrument for measuring such areas is called a *planimeter*, and for a full description of these the reader is referred to Brough's *Mine Surveying*, pp. 159, 160.

Levelling.—In connection with colliery operations a good deal of levelling is often required, both on the surface and underground. Levelling is defined as "the art of finding the difference between two points which are vertically at different distances from a plane parallel with the horizon."

To find the difference of level between any number of points three methods may be adduced :—(1) Trigonometrical; (2) physical; and (3) geometrical.

In *trigonometrical levelling* the lengths and angles have to be measured, this being usually accomplished by a theodolite, but it is a method not often resorted to, except under exceptional circumstances, such as when ascertaining inaccessible points on steep mountain sides.

Physical levelling is based on the change of atmospheric pressure at different altitudes from the centre of the earth, and is found by means of the aneroid barometer, which, as is well known, records the varying atmospheric pressure at different heights. With an ordinary barometer the mercury column falls on ascending hills on an average about 1 in. for every 900 ft. of ascent. This method is never practised for ordinary levelling, but it is exceedingly useful and much used by surveyors for ascertaining great altitudes where ordinary levelling would be impossible, or in making rough preliminary surveys in new and strange lands.

Geometrical levelling is the method most commonly employed by surveyors, and can be carried out with the different levelling instruments in use, or by the ordinary 'spirit level,' which forms a part of all such instruments.

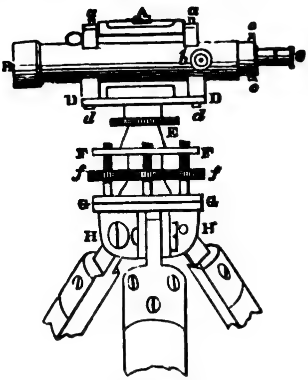


FIG. 505.

The instrument most largely used in Great Britain is that known as the 'Dumpy Level.' Fig. 505 shows the construction of this instrument. * A is the spirit level attached by screws at *a a* to the telescope B C. The small circle near the object end B of the telescope represents a small transverse spirit level used to show whether the cross wire of the telescope is truly horizontal. D D is a flat bar or oblong plate fixed on the top of the vertical axis E. To this bar the telescope is attached by adjusting screws *d d*. The hollow vertical axis turns upon a spindle fixed to the upper parallel plate F, the spindle being continued downwards, and being attached to the lower parallel plate G by a ball and socket joint. There are four levelling screws *f*, by which the vertical axis is set truly vertical. The lower plate is screwed on the tripod head H. The tripod consists of three wooden legs similar to those on a theodolite. In some instruments a compass is carried on the top of the plate *d d* for taking the bearings of lines of trial sections. The telescope is similar to that of the theodolite, except that the diaphragm contains one horizontal wire and two parallel vertical wires, as shown in fig. 506. It is usually 9 to 14 in. in length, the lines within the telescope being filaments of spider's web.

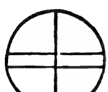


FIG. 506.

Adjusting the Level.—Before proceeding to level a section of ground the temporary adjustments of the instrument will have to be
ttended to each time the level is set up in a new position. The legs
the instrument should be firmly planted in the ground, and the
..llel plates made as horizontal as possible. The levelling of the
Fro.. nent is accon.. lished by placing the telescope with spirit level
chain li.. pair of the levelling screws, and adjusting the screws by
By referr.. oth in the one direction either inwards or outwards with
left hand b.. nd forefinger, till the bubble in the spirit level is brought
so that if we ..e centre; the telescope is now turned horizontally
this trapezoid = ..gle of 180° and put over the other pair of screws,
'evel adjusted as before. This may have to be
on with the others, four times until the bubble in the spirit level
avoid the division by ..centre with the telescope turned into any posi-
of all the trapezoids a.. ..eatise on Mine Surveying, p. 165.

tion. Fig. 507 shows the positions of the telescope during the adjusting process. Great care is required in levelling the instrument properly, and the operation should not be carried out too rapidly. The next operation is to adjust the telescope to prevent 'parallax,' that is, to move the eye-piece and to focus until the cross wires are seen with perfect distinctness.* Further adjustment may have to be made in sighting the levelling staff and repeated each time the staff is shifted. The nearer the object or staff the further the inner tube must be drawn out.

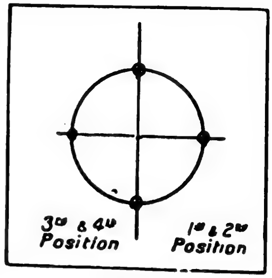


FIG. 507.

Levelling a Section.—Levelling is of two kinds, simple and compound. Simple levelling has only one line of collimation, whilst compound levelling entails constant changes of collimation. Simple levelling is resorted to when the difference in height between two points which are not far separated is required, and where one planting of the instrument will suffice. This is illustrated in fig. 508, where the instrument is placed as near as possible equidistant between A and B, the two points the difference of whose height requires to be determined. The telescope is first directed towards A, and a back

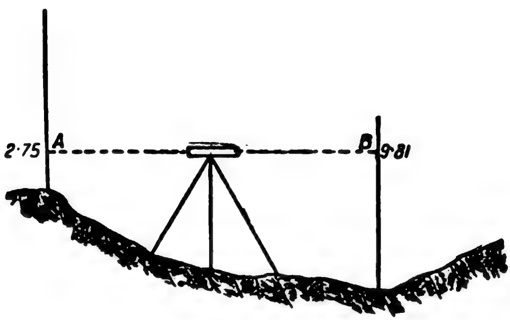


FIG. 508.

sight taken in which the line of collimation cuts the staff at 2·75; the telescope is then turned round and directed to B, and a foresight taken in which the line of collimation cuts the staff at 9·81; consequently there is a fall in the ground from A to B of 7·06 ft. When the foresights and intermediate are greater than the back sights the ground falls; if less it rises and will be deduced accordingly.

Compound Levelling.—When a long section of ground requires to

* Parallax means an *apparent change* in the position of an object caused by a change in the position of the observer.

be levelled, the undulations of the ground must be followed by varying lines of collimation, according to the rise or fall in the section.

This method of levelling is illustrated in fig. 509. The instrument is placed between A and B, and the reading of the back sight to the staff is taken; the telescope is then turned and a foresight taken to B. The instrument is now moved forward to a fresh position between B and C and a back sight taken on to the staff B at the same position as that at which the last foresight was taken; the telescope is again turned and a sight taken to C. The instrument is again shifted forward between C and D and a new line of collimation obtained, the same process being repeated till the whole section is levelled.

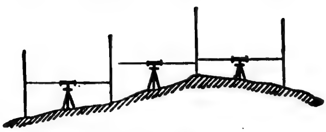


Fig. 509.

Datum Line.—In levelling, a datum line is generally selected for reducing the levels taken by the instrument to one fixed standard or relative height above sea-level. The Ordnance datum is "the approximate mean water-level at Liverpool," and all levels marked on Ordnance Survey plans are the altitudes in ft. above this datum. It is not, however, necessary or usual to adopt the Ordnance datum. Any height above the Ordnance datum may be selected, but it is best to select the datum sufficiently low to be below the lowest point in the section levelled, so that in plotting the levels will all be positive.

Booking the Levels.—In booking the levels various methods are employed, but the commonest and best method is as follows:—

Back Sight.	Inter-mediate Sight.	Fore-sight.	Rise.	Fall.	Reduced Levels.	Distance.	Remarks.
					Datum.	Links.	
9·6	...	...	...	...	100·00	00·00	From centre of bridge.
...	8·30	...	1·30	...	101·30	100·00	
...	7·50	...	0·80	...	102·10	200·00	
...	6·32	..	1·18	0·16	103·28	300·00	
...	6·48	...	...	...	103·12	400·00	
...	5·63	...	0·85	...	103·97	500·00	
...	4·20	...	1·43	...	105·40	600·00	
...	3·20	...	1·00	...	106·40	700·00	
...	...	2·21	0·98	...	107·39	800·00	
8·70	7·10	...	1·60	...	108·99	900·00	
...	6·23	..	0·88	...	109·86	1000·00	Opposite engine house.
...	4·35	...	1·88	...	111·74	1100·00	
...	3·45	...	0·90	...	112·64	1170·00	Opposite shaft.
...	3·50	...	...	0·05	112·59	1200·00	2 ft. from outside rail.
...	3·30	...	0·20	...	112·79	1300·00	
...	3·20	...	0·10	...	112·89	1400·00	
...	...	2·25	0·95	...	113·84	1460·00	7 ft. from outside rail.
18·30	...	4·46	14·05	0·21	113·84	...	
4·46	...	...	0·21	...	100·00	...	
13·84	...	...	13·84	...	13·84	...	

This method of booking levels gives more accurate results than any system. It may, however, be more compressed, as the work can be tested when the levels are reduced ready for plotting. If the sum of the back and foresights are added up, and the sum of the rises and falls also added up, then the difference between the sum of the back sight and foresight ought to be the same as the difference between the total of the rises and falls. The difference between the last reduced level and the original datum should also coincide with these two results, as shown in foregoing example.

Another method of booking the levels may here be shown, in which only four or five columns are used. This system of booking is employed when rapid results are required on the ground, and can be done very quickly when practised regularly, but is not a method to be recommended for students in general, on account of liability to error.

Instrument Height.	Sights.	Reduced Levels.	Remarks and Distances	
		Datum.		
62·75	12·75 B	50·00	On floor of machine house.	
	8·10	54·65	On level of discharging platform.	
	11·25	51·50	At peg.	0
	8·70	54·05		73
73·61	1·35 F	61·40		100
	12·21 B	61·40		100
	6·80	66·81		120
	4·50	69·11		200
	4·40	69·21		272
	3·45	70·16		303½
79·79	1·87 F	71·74	Fence	308½
	8·05 B	71·74		308½
	6·10	73·69		400
	5·70	74·09		500
	6·70	73·09		600
	8·50	71·29		700
	11·65	68·14		800
	12·00	67·79		820
70·49	13·10 F	66·69	Loading bank.	
	3·80 B	66·69	„ „	

In this method of levelling a datum is selected, as in the above at 50, the instrument is then directed to the staff and a back sight taken of 12·75, this is added to the datum, and will of course give the height of the instrument above the datum line.

Other intermediate sights are taken and recorded in the same column. These are deducted from or added to the height of the instrument, as the case may be, until a foresight marked F is reached; the instrument being now moved, another back-sight reading taken, and a new height obtained, and the reduced levels worked out as before.

Plotting the Levels.—When the levels are all reduced and ready for plotting, a horizontal base line A B, fig. 510, is drawn on the paper and the distances *a b*, *b c*, *c d*, *d e*, *e f*, etc., laid off to scale. At distance O, the first vertical line *a a* is drawn at right angles to A B, the height of this line being the selected datum line say 50 as an example. At *b*, or 73 from *a* (from example), the first reduced level (*b b*) is set up, and the distance 54·05 marked off. In the same way all the other reduced levels are set up until the section is completed. When plotting sections like this, it is usual to use two different scales for the horizontal and vertical distances, the vertical lines being drawn to scale from 3 to 6 times greater than the horizontal distances. By doing so, the depths of cutting and amount of embankment are shown with greater clearness than if both were drawn to the one scale, and requirements can be ascertained with much greater accuracy.

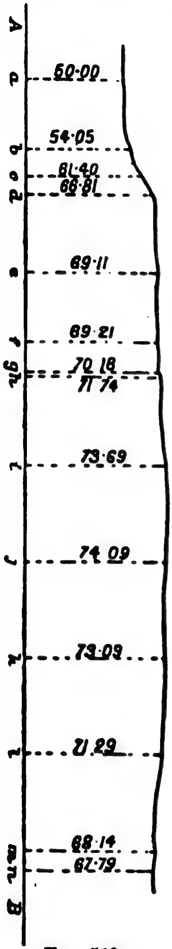


FIG. 510.

Surveying Problems. — The following worked examples are given, which may prove useful to the reader, more especially if he be studying for one of the mining examinations.

Question.—Find the area of a parallelogram whose sides are 7 and 15 respectively, and the included angle is 30° (fig. 511).

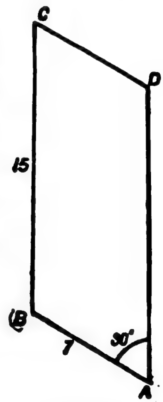


FIG. 511.

$$\text{Area} = BC \times DC \times \text{Sine } 30°.$$

$$\text{Sine of } 30° = 0·5000.$$

$$\therefore \text{Area} = 15 \times 7 \times 5 = 52·5 \text{ Ans.}$$

Question. — Find the ratio of the sides of a square to its diagonal. Find the length of side of square if the diagonal is 10 (fig. 512).

(1) The ratio of the side of a square to its diagonal is as

$$1 : \sqrt{2} \text{ or Ratio} = \frac{\text{side of square}}{\text{diagonal}} = \frac{1}{\sqrt{2}}.$$

$$\therefore \text{Diagonal} = \sqrt{2} \text{ side.}$$

(2) If the diagonal of a square is 10, the side of square $= \frac{10}{\sqrt{2}} = 7·07$ Ans.

Question.—A, B, and C denote the lengths of the sides of a triangle; how may its area be determined in terms of its sides?

$$\text{Let } S = \frac{A+B+C}{2} \text{ then area} = \sqrt{S(S-A)(S-B)(S-C)}.$$

Question.—The sides (A B, B C, C D, and D A) of a field measure 28, 45, 60, and 57 yards in length respectively. The angle ABC is a right angle. Find the area of the field in square yards (fig. 513).

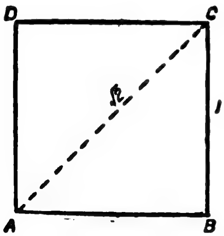


Fig. 512.

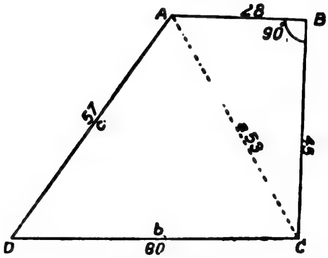


Fig. 513.

$$\text{The area of the triangle ABC} = \frac{28 \times 45}{2} = 630.$$

$$\text{The length of AC} = \sqrt{AB^2 + BC^2} = 53.$$

$$S = \frac{53+60+57}{2} = 85.$$

and the area of the triangle ADC =

$$\sqrt{S(S-a)(S-b)(S-c)} = \sqrt{85(32 \times 25 \times 28)} = 1879 \cdot 8$$

The total area of the field, therefore, equals 1379·8 + 630 = 2009·8 sq. yds.

Question.—The shape of a coalfield is rectangular, the sides 1 mile and ½ mile respectively. It contains three workable seams of coal of the following thickness and specific gravity.

1st seam, 3 ft. 0 in. thick. Specific gravity, 1·28.
2nd seam, 2 ft. 9 in. thick. Specific gravity, 1·29.
3rd seam, 2 ft. 6 in. thick. Specific gravity, 1·30.

Required the average available amount of coal in the field, after making a fair deduction for loss in working, there being no special pillars to be left in.

1 mile = 80 chains. ∴ area of field = 80 × 40 = 3200 sq. chains

$$1 \text{ acre} = 10 \text{ sq. chains.} \quad \therefore \text{area of field in acres} = \frac{3200}{10} = 320.$$

The total tons of mineral in a field = area × thickness of seam in in. × specific gravity × 100.

∴ total quantity in 1st seam = 320 × 36 × 1·28 × 100 = 1474560 tons
,, ,, 2nd ,, = 320 × 33 × 1·29 × 100 = 1362240 ,,
,, ,, 3rd ,, = 320 × 30 × 1·30 × 100 = 1248000 ,,

total quantity in three seams = 4084800 tons.

Allowing 15 per cent. for faults, bad coal, and loss in working, the total available amount will be $\frac{4084800 \times 85}{100}$ or 3472080 tons.

Question.—A seam dips 1 in 3; find the length of a level mine to cut a fault of 10 fathoms which intersects the seam (fig. 514).

An approximate solution, assuming the fault to be at right angles to the seam.

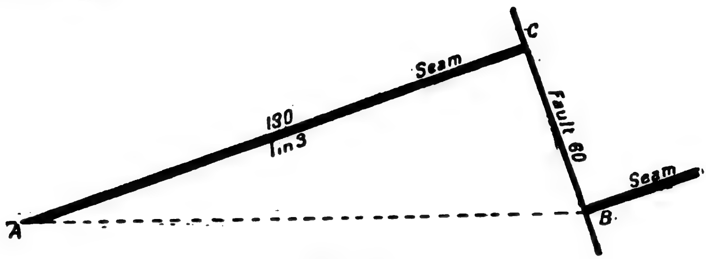


FIG. 514.

If the seam rises 1 in 3, what length of slope will give a rise of 60 ft.?

$$1 : 60 :: 3 = \frac{60 \times 3}{1} = 180 \text{ ft.} \quad \therefore AC = 180 \text{ ft.}$$

$$\therefore AB^2 = AC^2 \times BC^2 = \sqrt{36000} = 189 \cdot 7 \text{ ft. Ans.}$$

If in the above problem the mine rises 1 in 70, find what the length would be (fig. 515).

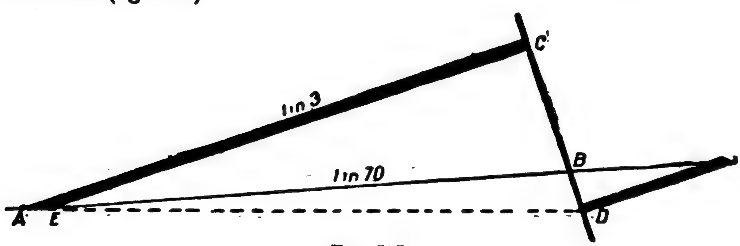


FIG. 515.

Let A D be an imaginary level line, and suppose we go along A D (A to E) 3 ft., then the line A C has risen 1 ft. above A D at E. The line A B has also risen $\frac{3}{70}$ ft. above A D. $\therefore$ in 3 ft. A B has parted from A C $1 - \frac{3}{70} = \frac{67}{70}$ ft.

If we gain $\frac{67}{70}$ ft. in 3 ft. how far must we go to gain 60 ft.?

$$\frac{60 \times 3}{\frac{67}{70}} = 188 \cdot 05 \text{ ft. } AC = 188 \cdot 05$$

$$\therefore AB^2 = (188 \cdot 05)^2 \times (60)^2 = \sqrt{38737 \cdot 80} = 196 \cdot 8 \text{ ft. Ans.}$$

Question.—A seam dips 1 in 7 towards the south. Level course is N. 70° E.; find the true bearing of a road rising 1 in 60.

Assume A C and A B to be of equal length (fig. 516). A C is level, and A B rises 1 ft. in 60 ft., therefore the point B is 1 ft. higher than

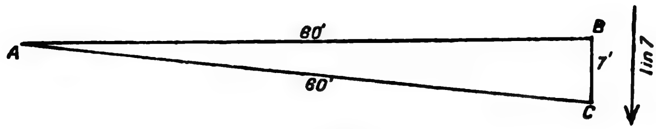


FIG. 516.

C. Now the seam is rising 1 in 7, and if it rises 1 ft. from C to B, distance C B must be 7 ft.

$$\text{By trigonometry sin C A B} = \frac{BC}{AB} = \frac{7}{60} = 0{\cdot}1166.$$

From tables sin 0·1166 = 6°·42′ and the bearing of road = 70° − 6° 42′ = N. 63° 18′ E. Ans.

Or let the diameter of the circle be 120 ft., then circumference = 120 × 3·1416 = 377 ft. nearly.

and 377 : 7 :: 360° = 6° 41′.

To find bearing

A B = N. 70 E.
∴ A B = N. 70° − 6° 41′ E.
= N. 63° 19′ E.

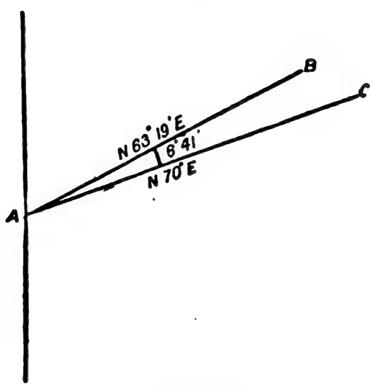


FIG. 517.

Question.—A seam is dipping 1 in 3 due south, a road is driven N. 45° W. Find the inclination of road. A road driven N. 90° W. will be level.

Sin 45° = 1·4142136
If A C = 100, B C also = 100 and

$$AB = \sqrt{100^2 \times 100^2} = \sqrt{20000} = 141 \cdot 42136$$

The seam is rising 1 in 3. $\therefore$ B is $\frac{100}{3} = 33 \cdot \dot{3}$ ft. above C and also $33 \cdot 3$ ft. above A; so that A B rises $33 \cdot 3$ ft. in $141 \cdot 4$ ft. and inclination $= \frac{141 \cdot 42}{33 \cdot 3} = 1$ in $4 \cdot 24$.

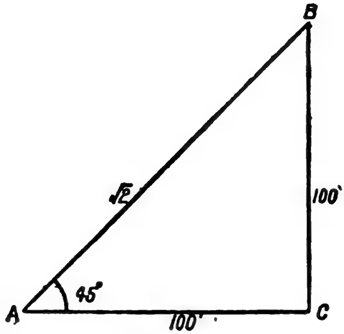


FIG. 518.

Question.—A seam is dipping 1 in 5 due south. A road is driven N. 60° W. Find its inclination.

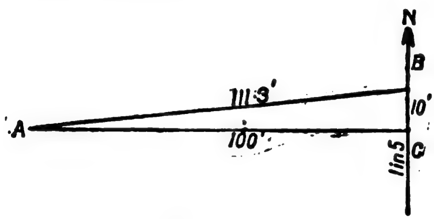


FIG. 519.

If A C = 100, B C = 50. $\therefore$ $AB = \sqrt{100^2 \times 50^2} = 111 \cdot 3$, if B C = 50 and is dipping 1 in 5 then B is $\frac{50}{5} =$ 10 ft. higher than C or than A, so that A B rises 10 ft. in $111 \cdot 3$ and $\therefore$ inclination $= \frac{111 \cdot 3}{10} = 1$ in $11 \cdot 3$ Ans.

Question.—A road has been driven 1° off the bearing. Find how much it is out in 500 ft.

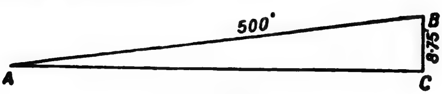


FIG. 520.

By trigonometry $BC = AB \times \sin BAC = 500 \times \cdot 0175 = 8 \cdot 75$ ft. Ans.

An approximate method of working a question of this kind, and

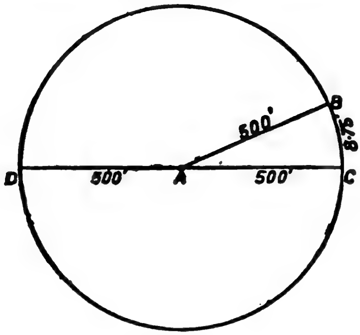


FIG. 521.

which may be useful to students who have no acquaintance with trigonometry, may be explained as follows :—

Let B C D be a circle with a radius of 500 ft. (fig. 521).

$\therefore$ The circumference of this circle $= 2 \times 500 \times 3 \cdot 1416$

$= 3141 \cdot 6$ ft.

Suppose the arm A B is swung round a complete circle it will have described an angle of 360°, and the point B will have travelled 3141·6 ft. Now, by proportion, if B travels 3141·6 ft. for 360°, how many ft. will it travel for 1°?

$$\therefore 360^\circ : 1^\circ :: 3141 \cdot 6 \text{ ft.}$$

$$\text{or } \frac{1 \times 3141 \cdot 6}{360} = 8 \cdot 72 \text{ ft. Ans.}$$

This distance, of course, is measured along the arc of the circle, but in a case like this the difference between the length of the arc and that of its chord is infinitesimal, and would be negligible for all practical purposes.

Question.—If a road has been driven for 200 yards and A is then found to be 30 yds. off the true course, which must have been the error in bearing in setting out?

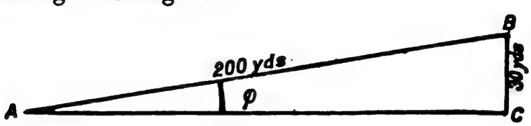


FIG. 522.

$$\text{By trigonometry } \sin \theta = \frac{BC}{AB}$$

$$= \frac{30}{200}$$

$$\therefore \sin \theta = \cdot 1500$$

$$\therefore \theta = 8^\circ\ 38' \text{ Ans.}$$

or by the approximate method mentioned above we have a circle whose radius is 200 yds., $\therefore$ circumference $= 2 \times 200 \times 3{\cdot}1416$
$= 1256{\cdot}64$ yds.

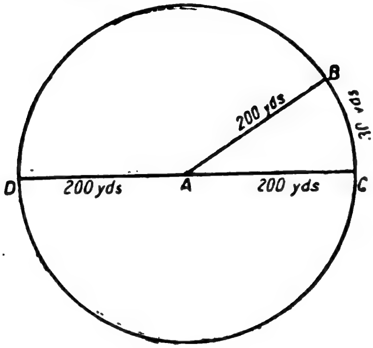


Fig. 523.

But the circumference is also $= 360°$.

$$\therefore 1256{\cdot}64 \text{ yds.} : 30 \text{ yds.} :: 360°$$

$$\therefore \frac{30 \times 360}{1256{\cdot}64} = 8° \ 35' \text{ Ans.}$$

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